REVIEW OF LITERATURE ON CONDITIONING AIR FOR ADVANCEMENT OF HEALTH AND SAFETY IN MINES

PART III. METHODS OF CONTROLLING THE CHEMICAL AND PHYSICAL QUALITIES OF UNDERGROUND AIR

BY J. J. FORBES, SARA J. DAVENPORT, AND GENEVIEVE MORGIS

* * * * * * * * * * Information Circular 7528

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UNITED STATES DEPARTMENT OF THE INTERIOR
J. A. Krug, Secretary
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James Boyd, Director

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Part III. Methods of Controlling the Chemical and Physical Qualities of Underground Air

by

J. J. Forbes[1/], Sara J. Davenport[2/], and Genevieve Morgis[3/]

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INTRODUCTION

This circular is part III of a series of papers reviewing the literature on air conditioning in mines, with particular reference to the health, safety, and efficiency of miners.4/ It deals with theoretical and practical methods of conditioning air in mines to correct or alleviate the health hazards described in parts I and II and discusses the results attained by some of the systems now in use in various parts of the world.

The principal conditions to be corrected or alleviated are the chemical qualities of mine air (especially when contaminated with toxic and noxious gases and dusts) and the physical qualities (abnormal mine temperatures and humidities) that emanate from various sources in underground workings. Air motion and barometric pressure are physical qualities also to be considered.

From the standpoint of health, knowledge of the composition of the air, or atmosphere, in mines and other underground operations is of prime importance. Suitable control measures or plans of procedure to combat or eliminate the hazards from contaminants must be based upon accurate information obtained by the selection and use of proper methods of sampling and analyzing mine atmospheres (1).5/

4/ Harrington, D., and Davenport, Sara J., Review of Literature on Conditioning Air for Advancement of Health and Safety in Mines. I: Bureau of Mines Inf. Circ. 7001, 1938, 38 pp.
Harrington, D., and Davenport, Sara J., Review of Literature on Conditioning Air for Advancement of Health and Safety in Mines. II: Need for Air Conditioning Indicated by Physical Quality of Underground Air: Bureau of Mines Inf. Circ. 7182, 1941, 104 pp.

5/ Numbers in parenthesis refer to bibliography at end of circular.

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SUMMARY

Conditioning air is one of the most essential factors to be considered in any underground workings, not only from the standpoint of health and safety but also with regard to efficiency and productivity. Without modern air-conditioning, the war's great demand upon the copper mines could hardly have been met, as the mechanical refrigeration cooled off sweltering holes in less than a month, whereas in the past it was standard practice to blow air through the tunnels for 3 years or so until they were cooled.

This paper discusses methods of determining the chemical and physical qualities of mine air and their control. Briefly, the following points are presented:

I. Chemical quality of mine air as affected by gaseous and particulate contaminants.

1. Selection and use of proper methods of sampling and analyzing the gaseous and particulate contaminants.

2. Methods of controlling gaseous and particulate contaminants through adequate ventilation, proper methods of working, dust suppression by water, and wetting agents. Necessary equipment for wetting dust is suggested.

a. Classification of dust-sampling instruments. (Discussion of the impinger method for determining the number concentration of dust - advantages and disadvantages.)

b. Chemical, petrographic, and X-ray diffraction procedures used to identify the constituents of dust.

3. Explanation of Maximum Allowable Concentrations (MAC). (Table listing MAC of certain gases and dusts found in mines.)

II. Physical qualities of mine air that affect the health, safety, and efficiency of miners and need for their control are presented. They are: Temperature, humidity, motion, barometric pressure, compressed air, and fogginess.

1. Determination of air temperatures by:

a. Dry bulb
b. Wet bulb
c. Whirling hygrometer
d. Hair hygrometer

2. Discussion of dew point.

3. Measurement of radiation by thermopile used with a hypersensitive galvanometer and globe thermometer.

4. Determination of cooling power of air.

a. Katathermometer
b. Effective temperature scale

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5. Explanation of the following instruments:

   a. Barometers (mercury, aneroid, recording)
   b. Anemometer
   c. Velometer
   d. Water gage

6. Sources of heat in mines.

   a. Heat due to adiabatic compression
   b. Heat from rock surrounding the air passages
   c. Heat created by man
   d. Heat created by machinery
   e. Heat due to oxidation
   f. Heat caused by blasting
   g. Heat caused by rock movements
   h. Heat caused by friction of air

7. Cooling of mine air by:

   a. Ventilation
   b. Cooling plant
   c. Refrigeration
   d. Air-conditioning system

If more detail of any phase of the subjects discussed is desired, it is suggested that the reader consult the bibliography at the end of this circular.

METHODS OF DETERMINING THE CHEMICAL QUALITY OF MINE AIR

For the purpose of this paper, contaminants that affect the chemical quality of mine air may be classified as gaseous and particulate.

After deciding on the purpose of the determination and the nature of the information desired, the most important steps are sampling and analysis.

Sampling involves careful selection of the proper place, time, and interval for making the determination or removing the contaminant.

The sampling of gases is ordinarily a much simpler procedure than the sampling of particulate matter, as gases diffuse more readily and do not settle significantly under the influence of gravity, and as their units or molecules are more uniform in composition and size than particles, which vary widely and the larger sizes settle under the influence of gravity; also the units of the latter tend to aggregate upon collision, thus changing in size and settling rate.

The concentration of gaseous contaminants ordinarily is determined by well-established chemical or physical procedures. The standardization of these determinations has progressed much farther than that of determinations for dispersoids (particulate matter).

Estimation of concentration is the main determination made on dispersoids. These estimations may be classified into (a) direct estimations, in which the

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concentration of the dispersoid is estimated directly by weighing or counting the number of microscopically visible units in samples collected from measured volumes of aerosol, and (b) indirect estimations, in which some property related to concentration is measured (2).

Determination of Gaseous Contaminants

Sampling

Gases that issue from rock strata, that may be produced in the normal course of mining, or that result from extraordinary occurrences, such as fires and explosions, may create health hazards that must be controlled. Only by the selection and use of proper methods of sampling and analyzing the mine atmospheres can the extent of such hazards be determined.

Although portable detecting devices such as the flame safety lamp, methane detectors, and carbon monoxide detectors may be used for examining mine atmospheres, all have definite limitations in their range of detection. Moreover, they are designed primarily for the detection of a single gas, whereas in many instances a complete analysis or determination of all the constituents of the mine atmosphere is necessary.

Methods of sampling gas may be divided into two general classes, which characterize the rate at which the samples are taken. These classes are termed "grab" (meaning instantaneous sampling) and "continuous" samples.

Grab samples are taken over short periods of time, say a few seconds to a minute or two, and represent the mine atmosphere at some particular place and instant. These constitute the majority of samples taken in mines.

Continuous samples are collected over a longer period than grab samples and usually are taken where the composition of the stream of gas to be sampled may vary from time to time. A sample taken continuously throughout the period will give the average composition. It is seldom necessary to take continuous samples in mines, as a number of grab samples taken at predetermined intervals generally will provide more information than one continuous sample (1).

Analysis

The most common procedure for analyzing gas samples is termed the "gas-volumetric" method, in which the mixture of gases is subjected to a series of treatments, each of which is designed to remove one particular constituent of the mixture without changing the proportions of the other gases remaining to be determined. To effect these successive removals, the gases are absorbed by reaction with suitable chemical reagents or, in the case of combustible gases such as methane, by combustion with air or oxygen. The procedure is started with a measured volume of sample in the analytical apparatus, and after each treatment the reduction of volume caused by removal of one constituent is measured. From these various volume measurements the percentage composition of the gas sample is calculated. The results of such an analysis usually are reported on a "percent by volume" basis rather than on a weight basis (1).

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The three types of gas-volumetric apparatus used by the Bureau of Mines for analysis of mine atmospheres are the Haldane (3), the laboratory Orsat (4), and the portable Orsat (5).

The particular field of application as well as the particular limitations of each of these types are discussed in detail in Miners' Circular 34 (1).

Determination of Particulate Contaminants

Sampling

Ordinarily in dust studies in mines the aim of dust sampling and determination is not to measure the absolute dust concentration in the air but rather to obtain an index of the health hazard involved in breathing the dust-laden air (6).

Dust-sampling instruments have been classified, according to physical basis of operation, into six groups: Settlement, filtration, washing, impingement, electric precipitation, and thermal precipitation. An extensive list of references to articles discussing the methods of using these various instruments in dust determination is given in Bureau of Mines Information Circular 7210 (2). Only the impingement method will be discussed briefly here, as it is the one used generally in the collection and determination of number concentration of dust particles in air in the United States by the Federal Bureau of Mines and the United States Public Health Service.

The impinger was developed by the Bureau of Mines in cooperation with the United States Public Health Service in 1922 (7). The underlying principle of this instrument is based on impingement and wetting of the dust particles by drawing air through a nozzle at high velocity onto a smooth surface under a bubbling column of liquid. The dust particles are retained in the liquid. The apparatus has been used widely, and permissible limits of dustiness have been established in the United States on the basis of results obtained with this device in correlation with medical studies. However, it has been criticized for its bulk, weight, and power requirements.

It was thought that these criticisms could be overcome to some extent by developing a more readily portable and easily operated instrument. It was also believed that a smaller instrument identical in principle and using the same counting technique should give the same results within experimental limits.

With these ideas in mind, the midget impinger dust-sampling apparatus was developed by the Bureau of Mines and described in Report of Investigations 3360 (8). The device is self-contained and easily hand-operated, and the complete apparatus, including 6 to 10 impinger flasks in a convenient case, need not weigh more than about 10 pounds.

The operation of the midget impinger is identical with that of the large impinger, except that a vacuum of 12 inches of water is required, compared with 36 for the large impinger. With a 1-mm. orifice and a vacuum of 12 inches of water, the flow through the instrument is approximately 0.1 cubic foot per minute.

Any satisfactory means can be used for drawing the air through the impinger. The suction for the first midget impinger designed was supplied by a small rotary pump of the vane type operated by a hand crank through a set of gears with a ratio of 9 to 1. A check valve and surge tank were inserted between the pump and impinger to obtain a smoother flow.

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Subsequently, a pump was designed especially for use with the midget impinger and gave excellent performance. It has four cylinders disposed radially at 90° about a single-throw crank. The intake valves connect to a collector ring, which in turn connects through a check valve to the first surge tank, which in turn connects to a second through a needle valve. A vacuum gage is used to indicate the suction necessary to pull through 0.1 cubic foot of air per minute. The pump is so designed that minor variations in the crank speed do not affect the flow of air significantly.

Number Concentration

The procedure used by the Bureau of Mines in determining, by the impinger method, the number concentration of dust in industrial atmospheres known or suspected to contain pneumoconiosis-producing material is described in Information Circular 7026 (9).

A microprojection method for determining the particle-size distribution and number concentration of dust samples collected by the impinger was developed by the Bureau of Mines (10) to obviate or reduce the eyestrain consequent to the regular filar-micrometer method and to save time and work. This method gives essentially the same results as the regular microscopic method, causes relatively little eyestrain, and is more rapid, in that it permits the counting of more concentrated samples without secondary dilution.

British investigators (11) have discussed the advantages and disadvantages of the impinger as follows:

> As a sampler, the impinger possesses some not inconsiderable advantages:
>
> (a) The sampling rate is high. This enables dust to be collected from a large volume of air, the importance of which has already been stressed.
>
> (b) It is comparatively easy to sample over a long period without undue multiplication of samples, even when the concentration of dust is high.
>
> (c) Evaluation of the samples is not necessarily restricted to microscopical examination. Separation of the liquid medium, either by evaporation or filtration, enables the dust to be estimated gravimetrically by the application of a micro-weighing technique. In cases where microscopical examination is also practised, this provides a valuable independent check on the gravimetric concentration computed from the size distribution. Alternatively, the concentration of dust may be expressed from optical measurements, either of light-extinction or of scattered light.
>
> The chief drawback of the impinger method of sampling is that the collection of small particles, which may be of importance from the health point of view, is far from absolute, although it is probable that the efficiency could be improved by increasing the velocity of impingement. The recognized low efficiency of the impinger for the collection of small particles is probably the chief reason why the samples yielded by it are commonly examined by light-field low-power technique only, where the lower limit of visibility is of the order of 1 m. A further defect of this method is that it is essentially an instrument which samples at a fixed rate.

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On the operational side, the disadvantages of the method are those which it shares in common with the gravimetric filtration methods, namely, that it is not self-contained, in the sense that it entails the provision of an independent source of power, and that the metering of large volumes of air over long periods presents some difficulties.

So far, little experience has been gained in the use of the impinger for sampling in coal-mines in this country, but its undoubted attractiveness in the directions which have been indicated, and the fact that the midget impinger has now been adopted as the standard instrument for dust-sampling in the United States, suggest that its possibilities are worthy of further exploration and test under British conditions.

Analysis

The various procedures that have been used to identify the constituents of dust include chemical, petrographic, and X-ray diffraction methods. Chemical methods are unsatisfactory owing to the smallness of the samples usually available and to the limitations and complexities of present methods. X-ray technique requires expensive apparatus not ordinarily available except in research laboratories.

As the petrographic examination of atmospheric dusts is simple and rapid, the Bureau of Mines (12) worked out and uses a method for determining a maximum value for the amount of quartz, which is the most important substance, from the standpoint of pneumoconiosis, in a dust sample. The sample is studied under a microscope, using dark-field illumination and immersing it in an oil with an index of refraction of 1.56. The material identified as similar to quartz may be anything from all quartz to no quartz. This method may give a high value for quartz, but not a low one, and if none is found there is only a negligible quantity in the dust. This method may be modified to identify other minerals by using oils of other refractive indices, and the same limitations will hold.

Maximum Allowable Concentrations

The fundamental purpose of applying maximum allowable concentrations is the promotion of health and efficiency; their practical use is for guidance in establishing control procedures to prevent harmful or objectionable concentrations of contaminants from accumulating in the air of working places.

Maximum allowable concentrations are an important tool of the industrial hygienist in the diagnosis of occupational disease when used in conjunction with careful clinical and physical examination and knowledge of the fundamental physiological effects of the materials for which they have been suggested.

As stated by Schrenk (13), all maximum allowable concentrations are subject to the limitations of measuring accurately pathological or physiological response and also to the limitations of the methods for sampling and analysis of atmospheric contaminants. From a consideration of these factors, it is apparent that it is not possible to establish permissible limits with a high degree of accuracy. However, this need not detract from the use of such limits, because the accuracy obtainable is satisfactory when permissible limits are used properly and are not employed for purposes for which they are not intended.

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Drinker (14) has explained briefly the general consensus of opinion in the application of MAC's to the evaluation of an occupational disease exposure and to the control of a health hazard. He states that the duration of exposure must be included. Most of the values presented today contemplate continued subjection to the atmospheric contaminant over the entire work week without injury to health.

If the work week is longer than normal, the concentration should be kept at a level lower than the accepted MAC, roughly in inverse proportion to the increase in time. However, if the exposure is for a period shorter than a normal week, great caution should be observed before permitting concentrations to be proportionally greater.

The extent to which the MAC can be exceeded, if at all, for short, repeated exposures depends on the characteristics of the substance. Where the physiological effect of the contaminant is largely irritative, as with a gas such as sulfur dioxide, the tolerable concentration for brief exposure is very little higher than that for longer exposure. Although an MAC of 10 parts of sulfur dioxide by volume per million parts of air is considered without effect for an indefinite period, a concentration of only twice this value would cause coughing in a short time.

Exposure to the silicosis-producing dusts can be greater than the accepted MAC's for short periods without injury to the lungs, but greatly excessive concentrations of even such a contaminant are to be avoided.

Where the exposure varies, the concentrations can be averaged within limits and again depending on the characteristics of the contaminant.

It is not unreasonable to average chronic exposures to substances like quartz. Thus, 4 hours' exposure to dust concentrations of 60 million particles per cubic foot and 4 hours to 20 might be considered equivalent to 8 hours at 40 million particles. That sort of averaging has been used in estimating silicosis risks.

Values for maximum allowable concentrations of some substances have been set by the American Standards Association and other organizations, as the United States Public Health Service and the Committee on Threshold Limits of the American Conference of Governmental Industrial Hygienists have suggested maximum allowable concentration values for a number of toxic and noxious substances that may contaminate the air of working places. Some States have issued regulations defining the allowable limits of certain substances in industry, whereas other States have suggested limits for possible use in controlling industrial conditions potentially hazardous to health.

Following are threshold limit values for gases, vapors, toxic dusts, fumes and mists, and mineral dusts that may be present in mines that were adopted at the April 1948 meeting of the American Conference of Industrial Hygienists:

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TABLE 1. - Maximum allowable concentrations of certain gases and dusts found in mines

Substance	P. P. M. 1/
Carbon dioxide	5,000
Carbon monoxide	100
Sulfur dioxide	10
Hydrogen cyanide	10
Hydrogen sulfide	20
Nitrogen oxides (other than N_2O)	25
Arsenic	0.5
Lead	0.15
Mercury	0.1
	M. P. P. C. F. 2/
Dust (nuisance, no free silica)	50
Silica:	
High (above 50% free SiO_2)	5
Medium (5 to 50% free SiO_2)	20
Low (below 5% free SiO_2)	50
Slate (below 5% free SiO_2)	50
Total dust (below 5% free SiO_2)	50

1/ Parts per million.
2/ Million particles per cubic foot.

METHODS OF DETERMINING THE PHYSICAL QUALITY OF MINE AIR

The principal physical qualities of mine air that affect the health, safety, and efficiency of the miner are temperature, humidity, motion, and barometric pressure. Additional hazards to health and safety may result from compressed air and fogginess, especially in tunnels.

Mining officials and those who are preparing to become officials should be familiar with and know how to use safely and efficiently the methods of determining the physical quality of mine air and the instruments and devices that are employed in and around mines for this purpose.

Among these are instruments and devices used in determining the temperature, humidity, pressure, velocity, volume, and certain other conditions of ventilation; they include thermometers, psychrometers, water gages, anemometers, and barometers.

Determination of Air Temperatures

Dry-Bulb Temperatures

The temperature of the air (dry-bulb) is measured by the mercury thermometer. There are various types of mercury thermometers to cover a wide range of temperatures, including maximum and minimum thermometers. Clinical thermometers (which are small and accurate) frequently are used underground for suitable temperature ranges. Recording thermometers are used when continuous records are required. Underground temperature readings vary considerably over short-time intervals. The position in which the thermometers are held, near the sides or in the center of the airway, makes an appreciable difference in the readings, often as much as 1.0° F. The personal factor also leads to considerable divergencies in actual readings (15).

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Wet-Bulb Temperatures

The wet-bulb temperature is measured by an ordinary thermometer, the bulb of which is covered by a piece of muslin that can be saturated with water. The water on the muslin will be evaporated by air that is not completely saturated, and this will cool the mercury bulb. The extent of the drop in temperature will depend on the dryness of the air and the efficiency with which the water in the bulb sleeve is evaporated. The drop in temperature of the wet bulb will continue until a constant state is reached, that is, when the heat balance is exact, at the muslin surface, between the heat given up by the air to the water in the muslin and the heat escaping from the water in the sleeve through the evaporated molecules of water.

If the wet-bulb thermometer is left in still air ("unventilated") a blanket of saturated air is formed around the wetted muslin, that is, the rate of evaporation will be inadequate, and the wet-bulb temperature so observed will be too high. To remove this blanket of saturated air, the thermometer must be placed in a strong current of air (a velocity of not less than 600 feet per minute), or the ventilating current must be artificially produced by whirling the thermometer, as in the type known as the whirling hygrometer. Important features in this type of instrument are:

1. There are two thermometers, so that (nearly) simultaneous readings can be taken of both the dry- and wet-bulb temperatures.

2. The wet-bulb sleeve sucks up a constant flow of water, by capillary attraction, from a small reservoir, which is held by a clip and can be detached for filling as required.

3. The wet-bulb sleeve commonly used on the Witwatersrand mines is of light muslin. It is kept clean and is renewed frequently. Thick sleeves are unsuitable.

The disadvantages of the whirling hygrometer (known also as the "sling psychrometer") are:

1. The necessary interval between the time when the whirling is stopped and the time when the instrument is read. The error from this cause can be reduced with practice but may be appreciable.

2. The effect of radiation from the rock walls, though much diminished by the operation of whirling, is still not negligible.

3. Even under the best circumstances, complete efficiency of evaporation is not obtainable.

With an unventilated wet bulb, the standard wet-bulb readings are dependent on the difference between dry- and wet-bulb readings. There is still some doubt as to the exact effect of the difference ("gap") between dry- and wet-bulb temperatures on the ventilated wet-bulb temperature. Under reasonably constant conditions, such as in an upcast shaft, wet-bulb temperatures may vary by about $\pm$ 0.5° F.

Several other types of hygrometers have been developed. Some of these are stationery and not whirled; the air is drawn past the wet bulb at a velocity of 600 feet a minute, by suction, and the bulbs are silvered to reduce the effects of radiation (15).

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Hair hygrometers are particularly useful for providing a continuous record. The underlying principle of hair hygrometers is that the contraction of a hair held in tension is a function of the relative humidity of the air. Within limits, it may be said that on exposure to a fairly dry atmosphere the contraction is proportional to the logarithm of the relative humidity. The contraction of the hair is transmitted to a recording drum. In general, it is difficult to obtain consistent results, and the observations can only be accepted as approximations, particularly when the relative humidity is low.

Dew Point

If air containing moisture, but not saturated, is cooled, a temperature point will be reached when the air becomes saturated and the moisture begins to condense from the air. The temperature at this point is called the "dew point" of the air. It will be noted that the dew point is not the wet-bulb temperature, the basis for which is evaporation without external cooling. Owing to the external cooling, condensation always starts below the wet-bulb temperature of the air, at the temperature where the original vapor content of the air represents saturation.

The relative humidity can be determined from the dew point. It is the relative pressure at the dew point compared (as a percentage) with that at the original dry-bulb temperature of the air.

The result is the same as that determined by observing the dry- and wet-bulb temperatures, as the vapor pressure of the air at dew point is the same as in the original air at temperature T, dry bulb, and T^1, wet bulb (assuming that there is no change in the barometric pressure).

The principle on which these instruments are designed is the observation of the temperature at which dew is deposited on a surface that is being cooled. A highly polished metal surface is used, preferably silver (or gold or platinum), which has a high thermal conductivity, so that when cooled internally the external surface temperature of the metal responds immediately. The cooling medium is the evaporation of ether, which can be controlled carefully by an aspirator (of the tube and bulb type).

In the types most commonly used, the condensing surface is in the form of a thimble soldered to a glass tube. Inside the glass tube and thimble, which is filled with ether, is placed a sensitive thermometer and an arrangement of glass tubes whereby air can be drawn through the ether to evaporate it. By careful control of the aspirator the dew can be made to appear and disappear without any appreciable change in the reading of the thermometer.

The temperature of the thermometer at the moment of dew deposition is read through a telescope to avoid errors due to the proximity of the observer.

Radiation

Instruments have been developed to measure radiation; the laboratory instrument is rather delicate and consists of a thermopile used with a hypersensitive galvanometer. The thermopile is directed to all points on the sphere surrounding the point of observation, and about fifty readings are taken.

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A more robust instrument is the globe thermometer, similar to an ordinary mercury-in-glass thermometer, but with the bulb inside a hollow copper sphere, the surface of which is finished in dull black. Larcombe (15) states that its efficiency compares favorably with that of the most delicate laboratory instrument.

It has been pointed out that the ordinary mercury thermometer with polished glass bulb has an emissivity 90 percent of that of a perfect black body, and that it is extremely susceptible to the effect of radiant heat. To avoid erroneous readings (when, for instance, the rock surface is at a much higher temperature than the air), the bulb of the thermometer should be covered with a light metallic foil, which reflects 90 to 95 percent of incident radiation.

Determination of Cooling Power of Air

Temperature, humidity, and air velocity are the determining factors in the cooling power of the air; the wet-bulb temperature and the velocity of the air are the two major factors controlling the efficacy of the air in cooling the human body. The dry-bulb temperature is of comparatively minor importance, though it has a definite effect when the gap between dry- and wet-bulb approaches 25° or 30° F., especially if the air velocity is high, 600 feet a minute or over (15).

Katathermometer

According to Jeppe (15) the infinite range of combinations of temperature, humidity, and air velocity led to the development of an instrument that would combine these factors in one reading and could be used for comparative purposes. Hill (15) developed the instrument called the katathermometer for this purpose. The instrument is a glass thermometer with a bulb about 4 cm. long and 2 cm. in diameter and a stem 20 cm. long, which has two marks on it representing, in the ordinary instrument, temperatures of 95° and 100° F. In addition to the large bulb, there is a small reservoir at the bottom of the stem and another at the top of the stem. The colored alcohol, which fills the bottom bulb, is driven up the stem into the top reservoir by heating the instrument, and the time required for the alcohol to fall from the upper to the lower mark is measured. This gives the rate of cooling of the instrument over the range 100° to 95° F. On the stem of the instrument is a factor prefixed by the letter F. This factor is the number of units of heat lost per square centimeter per second by the particular thermometer in cooling from 100° to 95° F.

The instrument can be used dry or with a moist sleeve on the bulb. In the latter case, the wet kata cooling power of the air is obtained, which, in hot atmospheres, is approximately equal to the rate of heat loss from the moist skin of the human body per unit of area, but not exactly so, because there are many variable factors controlling the heat loss from the human body that are not produced in an instrument. The dry katathermometer is used for the determination of the velocity of the air.

In a discussion of a paper by Barenbrug (16) on mining at depth on the Kolar gold fields, Mossop (16) stated that although the wet cooling power is widely used as an index of the "severity" of environmental conditions, it is not an accurate index. Mossop (16) stated that:

> It seems that a more promising approach to the problem of assessing the "severity" of heat environments lies in the separate measurement of the four relevant variables, namely air temperature, wet-bulb temperature, air velocity, and mean radiant temperature, and their combination by means of a chart or charts into a single index.

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Determination of Atmospheric Pressure

Barometers

Normal barometric pressure at sea level is about 29.93 inches of mercury, or 14.71 pounds per square inch, or 33.9 feet of water column.

The heights of the column of mercury vary with the atmospheric pressure. Even at sea level the variation may be 29 to 31 inches of mercury, depending on weather conditions; at high altitudes, such as are reached by airplanes, the barometric pressure may drop to less than 10 inches of mercury, whereas in deep mines it may increase to 35 inches or more (15).

The instruments built on this principle comprise strongly constructed cylindrical vessels that are closed at the top and fitted with chamois leather at the bottom. The pressure of the atmosphere acts against this leather bottom. Scales are provided for observing the height (top of the meniscus) at which the mercury column stands. Recording instruments operated by clock mechanisms are available, which mark the variations of height on a suitable chart.

The mercury barometer (17) is a glass tube about 3 feet long, closed at one end, and filled with mercury. The mercury is held in the tube while the latter is inverted and the open end is submerged in a cup or vessel containing mercury. The tube is then supported in an upright position and the mercury will fall in the tube until the distance from the top of the mercury in the tube to the surface of the mercury in the cup is about 30 inches at or near sea level.

There is essentially a vacuum in the space above the mercury in the closed end of the tube; therefore, the force that holds up the 30-inch column of mercury is the pressure of the atmosphere. The height of the mercury column will vary as the atmospheric pressure changes with respect to height above or below sea level and as to the density or weight of the air; consequently, an increase in atmospheric pressure will cause the height of the mercury column to increase, and a decrease in atmospheric pressure will cause the reverse. Changes in atmospheric pressure may be caused by a difference in altitude or in weather conditions. At sea level the barometer may register 29 to 31 inches, depending on the temperature, humidity, and density of the air; but normally the sea-level barometric reading is about 29.92 inches, and is about 24.89 inches at an altitude of 5,000 feet above sea level.

A cubic inch of mercury weighs 0.49 pound; therefore, a column of mercury 1 inch high exerts a pressure of 0.49 pound per square inch, and a column 30 inches high exerts a pressure of 30 x 0.49, or 14.7 pounds per square inch, which is held up by the atmospheric pressure; this is taken as the standard for atmospheric pressure at sea level when the mercurial barometer stands at about 30 inches and the temperature is 60° F. and is known as 1 atmosphere. Under these conditions, a cubic foot of air weighs 0.0765 pound.

The aneroid barometer (17) comprises a metal case about the size and shape of a large, thick watch from which the air has been removed, with the face graduated to a scale ranging from about 25 to 31 to correspond with inches of mercury. The back is made of thin, elastic metal. The back cover springs outward as the atmospheric pressure decreases and bends inward as the pressure increases. The movement is very slight, but it is amplified by the mechanism of the barometer and transmitted to a pointer or hand that moves along the scale and indicates the atmospheric

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pressure. Aneroid barometers should be compared frequently with a mercurial barometer and calibrated if necessary.

Recording barometers (17) of the aneroid type are obtainable and should be used where a continuous record of atmospheric pressure is desired. In mining operations, the barometer is used in connection with the liberation of gas. Storm or other atmospheric conditions, accompanied by a lowering of the air pressure or atmospheric pressure outside of a mine, as indicated by a "falling" or a "low" barometer, tend to permit an increase in the liberation of methane. Such a condition also may release gas that is being held by the ventilating pressure on gobs or behind leaky stoppings of imperfectly sealed regions and allow it to enter active workings. A sudden falling of the barometer may indicate a hazardous condition from a greater inflow of gas into the mine openings, and this danger is greatest adjacent to open, poorly ventilated, abandoned regions or to sealed regions where the pillars are shattered or the seals are imperfectly made or maintained. A barometer, therefore, is a useful and valuable piece of equipment at a mine and should be read frequently; precautionary measures should be taken promptly when the barometer changes suddenly.

Determination of Velocity of Air Currents

The velocity of an air current can be estimated roughly by walking with the current holding a pendant handkerchief. If the handkerchief remains vertical, the velocity of the air is about the speed of walking - 250 feet to 300 feet a minute. The angle of the handkerchief will indicate whether the velocity is appreciably greater or less than 250 feet a minute (15).

A lighted candle may be used and the speed of walking adjusted to keep the flame vertical (15).

For low velocities of 15 to 20 feet a minute, smoke or powders may be used for determining roughly the velocity of the air by observing the time taken from the release at one point to the arrival at another point (15).

Anemometers

Various instruments of the anemometer type have been designed to cover the whole range of velocities from 8 feet a minute to 10,000 feet a minute. The modern anemometer (modern windmill type), generally used for velocities from 200 to 2,000 feet per minute, consists basically of a light vane wheel carrying a number of light blades or vanes which are set at such an angle that, when a current of air impinges on the blades, the wheel rotates at a rate proportional to that of the air current.

Revolutions are recorded on a dial, each revolution corresponding to a linear foot of movement of the air. The recording is accumulative, so that the instrument is held (or rather passed systematically over the cross-sectional area) for a definite time, measured with a stop-watch, to obtain the velocity of air flow.

Anemometers must be tested regularly for accuracy and calibrated. They are provided with correction charts (both negative and positive) for all determined velocities (15).

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When an anemometer is used, it should be held or placed in the air current, so that the air will flow through the anemometer from the back or side opposite the dials.

To obtain the most accurate reading for practical purposes in mining, the anemometer should be held at arm's length, as nearly perpendicular to the air current as feasible, and moved slowly and regularly up, down, and across the airway so as to register as nearly as possible the average velocity of air flow. It is necessary to move the anemometer to different portions of the airway, because the air velocity is very likely to be greater in the center than near the sides, top, and bottom. Although this method gives the most nearly accurate average results, if "check" readings are to be made periodically in the same airway, the most accurate comparison can be made by placing the anemometer in a definite spot in the airway and taking the daily, weekly, or other periodic reading at the same place.

To take a reading, the anemometer should be held or placed in the air current as described, the lever that operates the dial mechanism should be pushed into gear or to the "on" position, which releases the dial hands; the time should be noted carefully; and at the end of one or more minutes the lever should be pushed to the "off" position to stop the dial hands.

After the velocity of the air current has been determined, it should be multiplied by the area or cross section of the airway to determine the number of cubic feet of air flowing in the airway. For example, if the anemometer shows 300 feet velocity per minute in an airway 6 feet high and 10 feet wide, the amount of air passing the point at which the reading was taken will be 300 x 60 (6 x 10 area), or 18,000 cubic feet per minute. More accurate readings may be obtained by adding or substracting certain constants or correction factors furnished by the maker for each anemometer. These correction factors, however, usually are of minor importance, as at best the anemometer only approximates the average velocity of an air current, owing to fluctuation of air current, variation in velocity in different portions of the airway, and variations in time and place of taking readings.

Usually, an anemometer will not function when the air velocity is less than 30 feet per minute, and readings under 80 to 100 feet generally are subject to correction and are likely to be erroneous (17).

The importance of low velocities under hot, humid conditions makes it necessary to have some accurate instrument for their determination. The Rees Torsion anemometer, based on the pendant-vane principle, is suitable for measuring air velocities of 10 feet to 180 feet a minute. A thin, rectangular aluminum vane is suspended from a horizontal wire. The ends of the wire are soldered into small brass nipples fixed in a frame. The vane swings out of vertical when a current of air impinges on it. The velocity of the air is obtained by measuring the amount of twist (torsion) of wire necessary to bring the vane back to the vertical position. The effect of the weight of the vane is eliminated, and the pressure of the air is always normal to the vane. An extension from the vane dips into an oil vessel so as to damp down its movement.

A correction for the density of the air must be applied, as with other anemometers.

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Yaglou (15), in 1938, described a heated thermometer anemometer capable of measuring accurately velocities of 10 feet to 6,000 feet a minute as follows:

> It consists of an ordinary glass thermometer with an electric coil surrounding its bulb. Small dry cells furnish the heating current, and the voltage is regulated by means of a rheostat. These auxiliaries are assembled in a small box.
>
> Readings to be taken are temperature of the heated and of an unheated thermometer and the voltage used. Velocity is read off a table or chart or computed from an equation.
>
> By varying the voltage, any velocity can be measured with accuracy.

The instrument automatically compensates for ordinary variations of temperature and is negligibly affected by humidity, radiant heat, and convectional currents of its own. It offers little or no obstruction to airflow, but requires correction for partial stem immersion when used in small pipes. It is particularly useful for measuring air movement in rooms, or in front of exhaust hoods that cannot be measured accurately by other methods.

Velometer

As a result of research work by Boyle (15) in the United States, a spring-controlled vane-type instrument, the velometer, has been developed and is now used extensively for the direct reading of air velocities in feet per minute. It has been possible to measure velocities as low as 1/10 foot a second by special arrangement of this instrument (15).

Water Gage

A water gage is used for determining differences in pressure or vacuum of ventilating air currents in mines (17). In its simplest form, the water gage is a vertical glass U-tube partly filled with water, both ends of which are open. An extension tube, which may be inserted into a small opening in a partition of the fan house or a stopping between the intake and return airways, is fitted to one end of the U-tube. A scale, graduated in inches or fractions thereof, is attached to the U-tube. The difference in inches between the height of the water level in one tube and the height of the water level in the other tube gives the water-gage reading regardless of how the scale is designed to be read.

Another type is the recording water gage, which gives a continuous chart record of the water-gage readings during a 24-hour period. Use of this type is highly desirable, particularly for fan-house installations, as it provides a continuous record of the ventilating pressure.

Much could be said about the importance and meaning of water-gage readings. The pressure registered by a water gage and the volume of air being circulated may indicate the condition of mine airways. A mine-ventilating system that requires an abnormally high water gage is generally inefficient and expensive.

High blood pressure in the human system is a sign of danger and indicates that the heart is being overworked. Similarly, a high water gage in mine ventilation

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indicates high power consumption in driving the mine fan, probably because there are not enough airways, they are inadequate in size, or they are choked with falls; in some cases probably all three may be true. Although the water gage generally indicates certain things with respect to the ventilation system, it cannot be depended upon to show everything that is caused by changing conditions. Many mine officials believe that the water gage will indicate all ventilating conditions; however, this is not true. Under some conditions a large fall may completely block an entry or entries or the air may be "short-circuited", yet the water gage will not be affected because the fan characteristics may be such that little or no difference will be recorded by the water gage.

It is necessary, also, to consider the amount of air flowing and the amount of power consumed by the fan motor when one interprets the record of the water-gage readings (17).

CONTROL OF THE CHEMICAL QUALITY OF MINE AIR

Need For Control

Data collected by the Bureau of Mines on the chemical quality of mine air in coal and metal mines in the various mining districts of the United States reveal that conditions in many of them are conducive to ill health and inefficiency of the workers.

When normal air, which contains about 21 percent (20.94) oxygen, is circulated through a mine it loses some of its oxygen through the breathing of men and animals; oxygen is consumed in mine fires and in explosions of methane or coal dust or both; a certain amount is absorbed by the coal and timber or combined chemically with minerals such as pyrite and marcasite; the oxygen thus absorbed is replaced largely by the carbon dioxide, which is given off by the coal and decaying timbers and by the breathing of men and animals; moreover, the oxygen content of the air may be diluted by intrusion of extraneous strata gases, such as nitrogen, carbon dioxide, methane, and hydrogen sulfide, or by gases formed by action of chemicals in mine water on minerals or rocks.

Atmospheres deficient in oxygen may occur in unventilated or poorly ventilated parts of all underground mines.

High oxygen depletion is common in metal mines, in abandoned timbered workings, where the air is still, particularly if much moisture is present. In reworking abandoned back-filled timbered stopes or other workings, miners complain of "timber gas"; this, by analysis, shows a depletion of oxygen up to several percent and the formation of CO_2 to about one-third of the amount of oxygen depletion. Air containing such small quantities of CO_2 as 0.3 percent and showing an oxygen depletion of 1.5 percent causes headache and discomfort to miners, especially if the air is still and the temperature is 75° F. or higher; possibly, however, the presence of objectionable odors or, if undetected, gases may have contributed (18).

An example of an atmosphere virtually without oxygen was demonstrated by a sample taken by a Bureau of Mines engineer wearing oxygen breathing apparatus in a Cripple Creek shaft some years ago during the recovery of the bodies of two victims of asphyxiation; analysis of the sample in the Pittsburgh laboratory of the Bureau showed the presence of 18.40 percent of carbon dioxide, 0.27 percent of oxygen, and

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81.33 percent of nitrogen, although the atmosphere appeared clear and normal. In a mine one can pass from good, breathable air to an atmosphere that will kill as quickly as one can dive from a board to deep water below (19).

Carbon dioxide does not support animal life. It is not poisonous and in small percentages in ordinary air stimulates breathing. It is formed by the breathing of man and animals, by the oxidation and decay of timber, and by the action of acid water on carbonates; it is also one of the gaseous products of blasting. Nitrogen forms approximately 79 percent by volume of normal air. When the oxygen of the air is burned or combines chemically with mine minerals and timbers, the nitrogen, being an inert gas, is unchanged. A man may be suffocated through lack of oxygen by both nitrogen and carbon dioxide, as in drowning (19).

The Bureau of Mines considers that any mine atmosphere normally worked in by man should contain at least 19.5 percent oxygen and not more than 0.5 percent carbon dioxide.

In Great Britain the CO_2 must be diluted with fresh air so as not to exceed a maximum of 1-1/4 percent, and on the Witwatersrand it must not exceed 0.2 percent (15).

As will be noted from table 1, the MAC (maximum allowable concentration) of CO_2 suggested is 5,000 parts per million.

Carbon monoxide is formed by the incomplete combustion of carbon or carbonaceous material. The main sources in underground coal mines are the spontaneous heating of coal and the explosion of methane or coal dust and in any mine underground fires, where combustion is incomplete, and explosion of blasting materials. In certain iron-ore mines of the United States, carbon monoxide was found in such percentages as 0.02, 0.04, 0.06, and even as high as 0.08. The MAC suggested is 0.01 percent (see table 1).

Sulfur dioxide is found in varying quantities up to one-fourth of 1 percent in fires in metal mines with sulfide ores and is to be expected in new fires in timbered sections when the fire has reached the incandescent stage (18). The MAC suggested for sulfur dioxide is 10 parts per million.

In some mines hydrogen sulfide issues from mine waters. It is produced by the spontaneous heating of coal containing pyrites, by the imperfect detonation of gunpowder, by blasting in heavy sulfide ores, and by the action of bacteria on timber or other vegetable matter in stagnant water. In low concentrations of more than 0.1 percent in the air, the sense of smell is deadened. It is extremely poisonous. The MAC suggested is 20 parts per million.

Oxides of nitrogen are formed in the detonation of explosives, especially when the oxygen balance of the explosive is not exact. Oxides of nitrogen are extremely dangerous, being considered even more poisonous than hydrogen sulfide. The MAC suggested is 25 parts per million. On the Witwatersrand the regulations provide that the ventilation shall be such that there is no determinable quantity of nitric or nitrous oxides (15)

Arsenical gases are found where enargite or other arsenical ores are exposed to fire. The MAC suggested for arsenic is 0.5 part per million.

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Failure to take proper action toward excluding fire fumes from other active workings of mines has caused many deaths and has greatly decreased the health and efficiency of workers, as minute quantities of CO, SO_2, and arsenical fumes lower vitality even if no permanent ill effects follow (18).

The most important chemical contamination of the air from the health standpoint is that due to the presence of dust, especially silica dust, although dusts of many kinds have come to be regarded as inimical to health (20). Even coal dust, which once was believed to be an "innocuous" if not a "protective" dust, is now considered as contributing to pulmonary abnormalites found among workers in certain coal mines. Investigations into the cause of the disease found among workers in the South Wales coal mines of Great Britain revealed that, although dust is the fundamental cause, the relative parts played by the different components of the dust are not yet elucidated and "the term selected should not prejudge the issue as to the exact causation of this condition" (20). The use of the comprehensive term "pneumoconiosis of coal workers" has, therefore, been recommended for all pulmonary conditions described as due to dust in workers engaged in any operation underground in coal mines (20).

It is the general opinion that the percentage of free silica in a dust is one of the measures of its danger and that, in general, men exposed to dust containing a high percentage of free silica contract silicosis more readily and rapidly than those exposed to a lower percentage.

Jeppe (15) states that:

> There is to-day however, an overwhelming mass of data that proves conclusively that the disease known as silicosis, in forms very similar to, if not identical with, classical silicosis, may be caused by dusts containing only a small percentage of free silica.
>
> The estimate of the "critical" percentage of free silica has been successively lowered from 50 percent to 40 percent; then to 30 percent; later to 15 percent or 20 percent, and finally to much lower figures, such as 5 percent or even 1 percent.
>
> Each year brings with it new examples of types of dust with low free-silica content causing fibrosis of the general silicosis-type, though in many instances the variations from classic silicosis are sufficient to enable investigators to determine by which type of dust they have been caused.

The MAC suggested for dust is as follows:

TABLE 2. MAC of dusts found in mines

Substance	M. P. P. C. F. 1/
Dust (nuisance, no free silica)	50
Silica:	
High (above 50% free SiO_2)	5
Medium (5 to 50% free SiO_2) ...	20
Low (below 5% free SiO_2).....	50
Slate (below 5% free SiO_2)	50
Total dust (below 5% free SiO_2)	50

1/ Million particles per cubic foot.

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Arbitrary standards for South Wales run-of-mine dust are summarized in table 3 (20). It will be noted that size of particles is taken into account in setting the standards.

TABLE 3. Arbitrary standards of dustiness for South Wales coal mines

Standard	Number of particles per cc.	
	1.0 to 5.0 microns, inclusive	0.5 to 5.0 microns, inclusive
Provisional	1,500	1,750
Target	800	1,050

Although these standards are being used to assess the degrees of dustiness in all mines in South Wales, the National Joint Pneumoconiosis Council (20) has distinguished between anthracite pits and other pits by recommending the following:

1. For coal-dust clouds in anthracite pits, 650 particles per cubic centimeter between 1.0 and 5.0 microns in size.

2. For coal-dust clouds in other pits, 850 particles per cubic centimeter between 1.0 and 5.0 microns in size.

In South African mines, where the dust is said to contain more than 85 percent free silica as quartz, a tentative standard of 300 particles per cubic centimeter of air (8.5 million particles per cubic foot) was set (21) as the upper limit of dustiness to be regarded as allowable. Rees (22) stated in 1947 that any konimeter count over 200 particles per cubic centimeter was considered high.

In a comprehensive study of dust problems in Pennsylvania anthracite mines from 1942 to 1945 by the Federal Bureau of Mines (23), concentrations of dust as high as 2,031 million particles per cubic foot were found in samples while undercutting without water, the average concentration being 581 million. Dust concentrations as high as 4,500 million particles per cubic foot were sampled during dry-drilling operations in coal and 1,688 million particles while dry-drilling in rock. The highest dust concentration sampled when coal was being loaded was 4,500 million particles per cubic foot, and the average of 123 samples was 753 million. Loading dry rock showed a high dust concentration of 1,875 million particles per cubic foot, with an average of 264 million particles for 18 samples.

Typical analyses of matter other than coal in air-borne dust in anthracite mines revealed the presence of 3 percent free silica (quartz) while loading, 58 percent free silica (quartz) while drilling bottom-rock hole, and 73 percent free silica (quartz) in sample of traction sand for motor haulage (23).

The Pennsylvania studies (23) also revealed that particle-size properties of air-borne dust at a magnification of 10,000 diameters of Owens jet, thermal precipitator, and midget impinger samples indicated an average particle size of 0.95 micron. A micron is 1/25,000 inch. Very few 10-micron particles were found; consequently, nearly all of the sampled dust is likely to be significant from the health standpoint, inasmuch as minus 10-micron dust is generally accepted as that most likely to be harmful to health upon being breathed.

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The examples given of dangerous concentrations of gases and dust in mine air indicate the necessity of controlling the chemical quality of mine atmospheres to protect the health and forward and maintain the working efficiency of underground workers.

Control of Gaseous Contaminants

Ventilation

Ventilation of underground workings consists in establishment of such control of air currents that the underground workers can work in safety with maximum comfort and efficiency and without impairment of health and requires that the mine openings may be made subject to such control of air flow as to remove from these workings, at ordinary times, harmful gases and dusts, and that at time of emergency, such as fire or explosion, there may be maintained as much or as little air flow as may be desired covering portions of the mine or the mine in its entirety (24).

In coal mining, the need for thorough ventilation of all underground working places is recognized by both mining practice and law. The laws of virtually all coal-mining States recognize good ventilation as a main safety requirement; the provisions of the laws on ventilation either agree or at least are similar in many respects (25).

Adequate dilution and removal of methane issuing or likely to issue from mine strata and removal of the carbon dioxide formed by coal and timber oxidation usually are provided in up-to-date coal mines by an air volume which, if adequately coursed, not only removes these gases but also the gaseous products of the detonation of explosives (which may include carbon monoxide, hydrogen sulfide, oxides of nitrogen, carbon dioxide, and nitrogen) and the carbon dioxide exhaled from the lungs of men and animals. Adequate coursing of air supplies the worker with air of normal (approximately 20.9 percent oxygen) quality, which moves with enough velocity and is adequate in volume to evaporate perspiration and usually is maintained at a temperature that will promote the worker's efficiency.

Ventilation is one of the most important single operations in mining, not only in protecting the health and lives of mine workers but also in promoting efficiency in operation and in lowering costs. Certainly, no other single phase of health and safety work affords greater protection to those employed in mines than an efficient ventilation system; on the other hand, lack of ventilation of the proper type is now and always has been the fundamental cause of much ill health and many accidents, particularly serious accidents, that afflict mine workers (25).

The Bureau of Mines (25) has recommended that every coal mine, large or small, should be ventilated mechanically by a fan on the surface. The fan should have sufficient capacity to give the workings an ample supply of air. It should be in fireproof housing and so arranged that the direction of air currents can be reversed with essentially no delay; the arrangement for reversing should be tested at least once a year. The fan should not be in front of the mine opening but should be offset, so that it will not be damaged by an explosion. Preferably, it should have two sources of power. At very gassy mines (those in which the return air from any split runs as high as 0.5 percent methane) it is desirable to have two fan units, one driven by one source of power and the other by a different source, one of these units

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being a "stand-by". The power line to the fan should be independent of the power lines that enter the mine. There should preferably be some automatic method of cutting off electric power to the mine when the fan slows down or stops. The fan should have a recording water gage and should be either constantly attended or have some automatic alarm to indicate any interruption or interference with the normal flow of air.

Mine air in which men work or travel should be improved when it contains less than 19.5 percent oxygen, more than 0.5 percent carbon dioxide, or is contaminated with noxious or poisonous gases. If the air immediately returning from a split that ventilates any group of active workings contains more than 1.0 percent methane, ventilation should be improved. Air with a carbon dioxide content of approximately 0.5 percent or an oxygen content of 19.5 percent very probably would not be harmful physiologically, yet would indicate sluggish or defective circulation of air.

The Federal Mine Safety Code (26) of standards and rules pertaining to safety conditions and practices in mines, dated May 29, 1946, agreed to by the Coal Mines Administrator and the United Mine Workers of America and subsequently adopted as part of the wage agreement between the U.M.W.A. and the bituminous coal-mine operators includes detailed regulations for ventilation and mine gases.

Metal mines have greater necessity to remove fumes from explosives, frequently have occasion to remove carbon dioxide, nitrogen, and other gases from strata, and even methane, the explosive gas commonly found in coal mines, is encountered occasionally (24).

Fan ventilation has been practiced in the mines on the Witwatersrand (15) since 1908, when it was started on the East Rand Proprietary Mines. Today, all except one or two shallow mines have main ventilating fans, and in addition auxiliary ventilation is provided in all development ends and in stopes (e.g., panel stopes and back stopes), where required.

In 1940, in 41 producing mines on the Rand, the total volume of air supplied, on the average, was 17,687,125 cubic feet per minute. Assuming the density of air (temperature, 60° F. wet; barometric pressure, 24.5 inches) as 0.06227, this volume of air represents 1,101,370 pounds, or 550 tons, per minute, that is, 33,000 tons per hour or 292,000 tons each 24-hours - about 4 tons of air, as an average, per ton milled. At the same time, the average volume of compressed air was 932,500 cubic feet a minute. The average volume of air supplied each 24 hours per rock drill in service was 1,868 cubic feet per minute.

For 1944, the total downcast in the 47 scheduled mines from Heidelberg to Klerksdorp was 19,915,400 cubic feet per minute, the compressed air (expressed as free air) being 940,400 cubic feet, giving a total of 20,855,800 cubic feet per minute, or 94 cubic feet per minute for each person underground (the maximum number of persons underground at any one time was 222,300).

The total number of main fans operating on the Witwatersrand runs into hundreds; on some mines there are 20 main fans; on many mines, 6 to 12; few mines have less than 2 fans. Sizes of main fans range from about 35 inches up to 30 feet in diameter. The fans commonly used circulate 200,000 to 400,000 cubic feet of air per minute against water gages of 2 to 3 inches. Some have capacities up to 500,000 to 600,000

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cubic feet of air per minute, and two or three a capacity of over 1,000,000 cubic feet per minute against water gages of 10 inches or more.

Various types of fans are used, including Propeller, Aerex, Screw, Aeroto, Sirocco, Keith-Blackman, Hohenzollern, and Walker.

For auxiliary ventilation, small compressed-air or electrically driven fans or "venturis" are used.

The present tendency is toward a still greater use and elaboration of fan equipment for improving conditions everywhere underground, with fans specially designed for particular purposes, and accompanied, when necessary, by air-cooling equipment (15).

Metal mines should provide fan ventilation from start of opening to avoid dangers from explosive fumes, dusts, and heat, and to supply at all times such volume of circulating air at places where men work as will enable them to exert themselves in comfort at maximum physical capacity without danger to health (24).

With the exception of blind-end working faces, metal-mine air in general is not particularly deficient in quality. However, blind-end faces of drifts, crosscuts, raises, winzes, and stopes in metal mines are likely to have air deficient in oxygen and high in nitrogen or carbon dioxide, and possibly in carbon monoxide, oxides of nitrogen, or other impurities from the use of explosives. There are many cases on record of asphyxiation in metal mines from these gases.

At seasons of the year when the temperature of surface air and of underground rock and water are about equal, mines relying on natural ventilation are likely to have periods when air circulation is sluggish or ceases utterly or reverses in direction.

During a mine fire, naturally ventilated mines are likely to be at a decided disadvantage through inability to control the direction of air currents.

There are records of naturally or otherwise inadequately ventilated mines filling with an atmosphere containing excessive nitrogen, carbon dioxide, or other gas, overcoming some of the workers (in some cases with fatal consequences), and compelling suspension of work for a long time. Upon establishment of efficient mechanical ventilation, such a situation can be controlled readily.

Whereas the costs of establishing ventilation systems for large mines are not uniform, the cost of operation is not particularly burdensome. The operating cost will almost invariably be offset by savings in compressed air and increased efficiency and health of employees, which frequently will offset the entire cost of the investment within a few years. If a fire occurs, an efficiently installed and operated mechanical ventilating system is of incalculable value, and the absence of such a safeguard is likely to result in heavy loss of property and possibly of life.

Control of Particulate Contaminants

Jeppe (15) calls attention to the fact that many mining fields have their individual problems in ventilation. Thus, in fiery coal mines a large volume of air is necessary to dilute the deleterious gases (and the coal dust), but in general mining

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methods are simple, depths are shallow to medium (coal mines over 4,000 feet in depth are rare), and multiple entries of ample cross section are economically practicable, so that splitting of the air current is characteristic of coal-mine ventilation practice, and thereby provision of large volumes of air is simplified.

In many mines, both base-metal and gold, there are many reefs and the workings are extremely intricate, forming a network extremely difficult to ventilate efficiently.

At mines in countries where intense cold prevails for the greater part of the year, it is often necessary in winter to heat the air before sending it underground.

At mines in very hot climates, or where the geothermic gradient (the rate at which the rock temperature rises with depth) is very rapid, large volumes of air have to be provided to maintain normally good ventilation underground.

Finally, in many mines dangerous dust is created in the course of mining, which must be diluted with large quantities of air, as in fiery coal mines.

Dusts (particulate contaminants) in metal-mine air are derived largely from dry drilling, blasting, shoveling or mucking, tramming or dumping rock or ore, timbering, and hauling.

Dust is created in coal mines through processes similar to those in metal mining, as well as by cutting, shearing, and mechanical loading. The dustiest, most unhealthful occupation underground is dry drilling.

The replacement of hand methods by mechanical means for coal production has greatly increased the proportion of fine dust in coal mines.

In the control of dust, the slogan should be "All dust is dangerous." No single measure of control is applicable to all dusty operations, and several methods of prevention and suppression have been practiced very successfully.

The reduction of dust underground depends on good ventilation practice, the adoption, as far as possible, of working methods that minimize the production of dust or the exposure of workers to the dust, and the use of dust-suppression measures (27).

Ventilation

The data on ventilation as a method of controlling gaseous contaminants in mine atmospheres are applicable, also, to the control of particulate contaminants. Dust concentrations can be reduced by improving ventilation. In coal mines, every practicable step should be taken to insure the best ventilation at the coal face. Excessively high air velocities should be avoided, however. When the air velocity exceeds about 300 feet a minute there is a tendency to raise settled dust. At still higher velocities, larger particles cause discomfort by impinging forcibly on the faces, particularly the eyes, of the workers (27). The Bureau of Mines' (23) study in the anthracite region indicates that the velocity of the air should not be less than 50 nor more than 400 feet a minute.

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Working Methods

Working methods may be used to minimize the production of dust or the exposure of the workers to dangerous concentrations. The dry "jackhammer" with conventional steel to drill coal or rock should be entirely supplanted by a dry jackhammer with spiral steel, wet jackhammers, air rotary drill with spiral steel, electric auger, or hand auger. To reduce the high concentrations of dust in blasting, ventilation should be increased, or the area should be wet before firing and when reentering. Having the workers wait a sufficient time for the dust to settle before reentering the blasting area will prevent their exposure to dangerous dust concentrations.

Results of investigations of the Bureau of Mines (28) show settling rate of dust after blasting under the following conditions:

1. The atmosphere was bratticed off for 3 hours while the dust samples were being taken. In this period, under these conditions, the dust concentration dropped from approximately 5,000 million particles per cubic foot to 50 million particles per cubic foot.

2. The atmosphere was bratticed off until 26 minutes after blasting, when the brattice was raised and natural ventilation (which was virtually negligible, except for eddy currents) was allowed to dilute the dust-laden air. It required 2 hours and 20 minutes after blasting for the dust concentration to drop from approximately 5,000 million particles per cubic foot to 50 million particles per cubic foot, and 2 hours and 50 minutes to drop to 15 million particles per cubic foot.

3. The atmosphere was bratticed off until 37 minutes after blasting, when the brattice was raised and 10-inch canvas tubing was installed to about 50 feet from the face. At 46 minutes after blasting, about 1,000 cubic feet of air per minute was forced into the face. It required 66 minutes after blasting for the dust concentration to drop to 50 million particles per cubic foot and 70 minutes to drop to 15 million particles per cubic foot.

The following table shows the time, in minutes, required for the dust concentration to drop from 825 million particles per cubic foot to 50 million and to 15 million particles per cubic foot under the three different conditions:

TABLE 4. - Time required for drop in dust concentration under certain conditions

Ventilating condition	Time, in minutes, to drop from 825,000,000 particles per cubic foot to -	
	50,000,000	15,000,000
No ventilation	121	181
Natural ventilation	105	135
Mechanical ventilation ..	20	24

Placement of boreholes and the amount of explosive used are important in preventing the formation of excessive dust during blasting.

Dissemination of harmful concentrations of dust by spillage during haulage of coal and rock can be prevented by proper maintenance of cars and care not to overload

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them. Substitution of nonsiliceous material for the highly siliceous sand used as traction material will reduce the haulage-dust hazard. Iron-ore tailings ground to sand size have been used as traction material for locomotives with apparent success.

Extremely dusty operations or working places that may affect adjacent workings can either be placed on separate splits of air or worked on off shifts. Several companies work alternate shifts in alternate working places to expose a minimum number of men in dusty atmospheres. Other companies limit four adjacent working places as the maximum for a separate split of air.

A vigorous effort should be made to keep roadways and travelways reasonably free of pulverized material, coal, and rock. Good housekeeping helps to reduce concentration of dust produced by the haulage of coal and rock and the passage of employees, as well as the number of injuries from stumbling and falling.

In addition to the use of large volumes of air to ventilate the mine workings, together with strict medical preemployment examination, the following operating practices have contributed to the success of the silicosis-control measures at Broken Hill, New South Wales, where "no man who has done no mining elsewhere and who entered the industry since 1922 has yet been found suffering from silicosis" (29):

1. Blasting only at end of shift, with 1-hour interval before beginning next shift.

2. Five shifts of 7 hours each a week.

3. The elimination of night-shift work in all stoping operations.

4. The elimination of all dry drilling.

5. Thorough wetting-down of all broken material after blasting and during barring-down and filling operations.

6. The enclosure of all ore passes and provision, where necessary, of dust-removal units and filters at tipping and loading points.

7. The running of ore passes at end of shift only where practicable.

8. Restriction of pneumatic stowage of filling, where this method is used, to night shift only, and arrangement of pneumatic-stowage operations so that dust from one operating locality does not reach areas where other men are working.

Dust Suppression

Wet Methods

Dust-collecting devices have been developed and have proved helpful in reducing the amount of air-borne dust in certain operations, but the most successful measure has been the allaying of dust by application of water.

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Drilling. - As dry drilling is the dustiest operation in mining, wet drilling will keep air dustiness within safe limits if used with care and judgment in collaring wet, using plenty of clean, dirt-free water and keeping the drills in good repair, with adequate regulation of the proportion of water and compressed air flowing through the drill steel (30).

In tests by the Bureau of Mines (31), the dust concentration in the air during actual drilling decreased as the water flow through the drill increased. The water flow for stopers was 0 to 3 gallons a minute and that for drifters from 0 to 1.02 gallons a minute. The rate of decrease for stoper drills was greater for water flows from 0 to 1.3 gallons per minute than for higher rates of water flow. The rate of decrease for drifter drills was similar to that for stoper drills at corresponding water flows and quite constant for water flows of 0 to 1.02 gallons per minute.

In most pneumatic rock drills, the compressed air passes through the drill steel with the water. Ordinarily, some of the high-pressure air supplied to the drill escapes into the back end of the drill steel, even when the operating throttle is not in the blowing position and travels up to the face of the hole being drilled. The amount of air escaping is affected by the flow of water through the drill steel; as the water flow through the drill steel increases, air flow decreases. Mining officials in South Africa concluded that such air leakage, although probably facilitating removal by the water of cuttings from the hole, was responsible for entrapping fine dust and blowing it into the air during drilling (32). Consequently, they devoted much attention to designing drills that permit the minimum practical air leakage through the drill steel and have established regulations on the use of such drills.

Tests also have been conducted by the Bureau of Mines (33) on the effectiveness of spraying water along the outside of the drill steel compared with that of standard wet-drilling practice. The tests show that spraying water along the outside of the drill steel should not take the place of regular wet drilling practice. However, if a method is developed by which the device may be used on holes of any inclination, it might prove to be a valuable accessory to regular wet drilling in collaring and drilling the first 2 feet of a hole. Tests show that the greatest amount of dust is thrown into the air of a working place in drilling the first 2 feet of a hole (34).

In hand-drilling coal, an average of a large number of tests indicates that about 35 million particles of dust per cubic foot of air is representative of this operation, although individual tests may show two or three times this concentration. In one mine representative samples averaged about twice this number for hand-drilling coal. Electric drilling of coal (based upon tests in about a dozen mines) will produce about 105 million particles of dust per cubic foot of air. Although the dust counts cited are relatively high, the time of drilling ranges from about 5 minutes for hand drilling to about 1 minute for electric drilling, and therefore the amount of dust thrown into the air is not excessive. Virtually all of the dust could be kept out of the air if a stream of water were directed into the collar of the hole during drilling.

Pneumatic-hammer drilling of rock, which is becoming increasingly important in mechanical mining, presents a real dust hazard. Only a few tests have been made with such drilling, but the results are arresting. In one mine, drilling in shale raised an average of 866 million particles per cubic foot of air, and in another mine

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1,135 million particles. In the same mine, where the bottom rock was drilled with electric auger drills the density was only 4.4 million. The answer to this problem is to use wet drilling or, where possible, to use electric drills.

Blasting. - The place (walls, floor, roof or top, and timbers) should be wetted thoroughly before blasting is done, and a water blast should be used during and after blasting. The region should be wetted thoroughly when the men return to the face region after blasting, and the muck pile should be kept well wetted at all times during loading, as the water not only lays the dust but also either absorbs or otherwise aids in diluting or eliminating harmful or poisonous gases that result from blasting and usually cling to the blasted material.

Tests (35) on the effectiveness of a spray of water mist in reducing the concentration of dust in the air after blasting and during mucking of the blasted material were conducted in three headings in medium-hard to hard rock. The use of the spray apparently made it possible to reduce the concentration of dust generated and thrown into the air during blasting more than 99 percent from the amount present when the spray was not used. Dust dissemination during mucking of material sprayed several hours before mucking was begun was much less than during shoveling of material thoroughly sprinkled just before and during mucking; and, of course, the dust present after use of either the spray or the sprinkling method is far less than if the mucking had been done dry.

Undercutting. - In an investigation by the Bureau of Mines (36) to determine the effect of using water on the cutter bar of coal-cutting machines, the quantity of dust raised into the air during undercutting ranged from 65.5 to 163.9 million particles per cubic foot of air, averaging 120.9 for the dry method. When water was used on the mining machines in the same working places, the dust ranged from 3.6 to 42.9, averaging 21.8 million particles per cubic foot of air. If dust raised during wet undercutting is taken as 100 percent, the average without water was 554.6 percent; that is, the operation became 5-1/2 times as dusty.

Top cutting. - The devices used to wet coal with top-cutting machines have been unsatisfactory (36). Whereas a reduction in about 71 percent resulted from wet undercutting, water applied to the cutter bar of the top-cutting machine resulted in a reduction of only 37 percent, and the density, even so, was 374 million particles per cubic foot of air. Comparable results with undercutting in another mine gave an average of 19.4 million particles of dust per cubic foot of air with wet cutting and 134.2 million particles with dry cutting. In the same mine, the average count for dry top cutting was 567.2 million particles and for wet top cutting 179.4 million particles of dust per cubic foot, a reduction of 72 percent with top cutting when using water but a concentration still too high for a healthful atmosphere. In these tests a 1/2-inch pipe drawn to an opening about 1/4 inch at the end was directing a stream of water onto the machine bits as they emerged from the kerf (37).

Under similar conditions, sharp bits produce less dust than dull bits. In fact, some tests conducted on this phase of the dust problem indicate that although the dust formed per cubic foot of air with sharp bits is only about one and one-half times as great as with dull bits, the longer time required to complete the cut with dull bits indicates that about twice as much dust is produced under similar conditions. The limited number of tests does not allow definite conclusions to be drawn, and further tests should be made to obtain more conclusive results (37).

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Loading. - Hand loading onto conveyors is generally done dry. The only operation studied in which water was used was one in which water had been applied at the end of the cutter bar of the undercutting machine, but the coal was loaded without further addition of water. The coal was 2 feet 10 inches thick and, after blasting, was loaded onto a shaker conveyor. The loading of wet-cut coal produced an average airborne dust density of 18 million particles and 50 million particles per cubic foot of air when cutting had been done without water. In each test the concentration of dust in air was less than 16 million particles during loading of most of the cut, but when the back of the cut was reached, and the fine coal left by blasting, together with machine cuttings not properly scraped from the cut, was loaded onto the conveyor, the counts increased to 26 and 47 million where cutting was done wet and 77 and 140 million particles per cubic foot of air where the dry method was employed. These figures and personal observations indicate that there is considerable advantage in cutting wet; for, although in some of the tests the wetting was not considered satisfactory from the point of view of allaying dust during cutting, the dust frequently was greatly reduced during loading.

One of the greatest dust hazards in mechanical loading lies in the fact that in some systems of loading by hand onto conveyors or loading by mobile loaders onto cars, large piles of fine coal are left in rooms, either from spillage or thrown back because of high sulfur or high ash in some of the coal or in the machine cuttings. The recommended method for overcoming this condition is to load out the dust and fine coal. It has been stated that it is not economically possible to load out this rejected coal; and although it is believed that no system should be adopted that jeopardizes the lives of employees, it is recognized that all mine operators will not accept this viewpoint. The hazard of leaving dust in the mine may be minimized if the piles left in working places are thoroughly wet, preferably with wetting agents, and a thick coating of rock dust is spread over the piles.

The coal pile should be kept wet during loading onto the conveyors, and near the end of each shift dust and loose coal that have accumulated along the belt or pans should be cleaned up, and the area should be rock-dusted. The walkway beside the conveyors should be kept rock-dusted or sprinkled, whichever method is best-suited to local conditions. Sprays should be installed at the discharge boom, and the coal should be wetted as it is loaded into the mine cars. As an additional precaution, the area around the loading head should be kept rock-dusted. A spray should be placed on the gathering side track, and the top of the loaded cars should be wetted before the trip leaves for the surface or dumping point (37).

Dust on mobile loaders is thrown into the air at the loading head and at the loading boom. Generally it is not possible to install nozzles on the loading boom, as the top of the boom frequently strikes the roof and cross bars, and the underside of the boom scrapes along the top of the cars. The end of the boom also is often buried in the coal as the car is loaded. Nozzles can be installed at strategic points on the loading head of most mobile loaders. Some tests have been made in which the coal pile was sprinkled before and during mechanical loading. In one mine the dust in an entry at a point where one cut had been taken in a room neck was collected from the air and found to have a concentration of 276 million particles per cubic foot of air when coal was loaded dry and 40 million particles when loaded when the coal was sprinkled. Tests in another mine gave averages of 139 and 44 million particles per cubic foot of air in dry and wet loading, respectively, the coal pile being sprinkled before loading begins and during loading if the coal gives off

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appreciable dust. These tests were so limited in scope that they are not indicative of mechanical loading where water is used. Nevertheless, the results so closely approximate tests made with hand loading under similar wetting conditions that the results possibly approximate average conditions when coal is loaded mechanically (37).

Sprinkling the coal by means of a hose during loading is not feasible in many mines where the block system of mining is followed, as the rooms and entries are generally so narrow that it is not safe for a man to stand beside the mobile loader. When nozzles are not installed on loading machines, suggested methods of wetting the coal pile during loading are:

1. Thoroughly wet the pile shortly before loading.

2. Place a spray nozzle on each side of the conveyor near the top of the machine so that sprays will be directed on the coal as it is loaded into the loading head.

3. Where this method is not feasible, or where crossbars are employed to support roof at the face, a pipe equipped with several spray nozzles could be attached to the front of the first crossbar and a hose attached to this pipe, so that water or other liquid could be sprayed on the coal as it is loaded. Where steel H-beams are used, suitable, readily adjustable clamps could be provided for quickly attaching the pipe to the bar. Suitable clamps for quickly fastening such a pipe to wooden bars may also be provided at little cost. If the application of water at the loading head is adequate, there should be little if any need for additional wetting at the discharge boom (37).

According to the experience of the Island Creek Coal Co. (38), 90 percent of the dust concentration at the coal face can be eliminated by continued spraying and good ventilation. The wet method used is an arrangement of sprays on the dust-producing equipment that will inject water, with a wetting agent, in an envelope enclosing the dust-producing area. The dust in suspension is attacked, and no attempt is made to wet the coal and prevent dust suspension. An average dust concentration of 300,000,000 particles per cubic foot was reduced to 30,000,000 particles per cubic foot.

J. A. Saxe, chief engineer, Island Creek Coal Co. (38), stated that it was necessary to operate all sprays continuously and effectively. The design used proved that spray stoppage can virtually be eliminated with little cost of attendance. In 1942 the Island Creek Coal Co. was loading 30,000 tons a day with mobile loaders, using track and shuttle-car equipment, with 35 spray units in use protecting virtually all the production. No time or production was lost by the use of the sprays. The spray systems were received enthusiastically by the workmen, who took unusually good care of the equipment and reported any interruptions in spraying.

Details of dust-prevention practices for some time in the Butte mines are, briefly, as follows (39):

> The roads and yards are oiled around the collars of the operating shafts or main inlet air courses.

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Water sprays and compressed air-water blasts are installed in the operating shafts at various elevations and are used to maintain moisture.

Water sprays with remote control are provided at skip-loading chutes, and station tenders are provided with respirators, though no high dust concentrations ordinarily occur during loading, because the ore has been wet previously in the mine.

Shaft stations must be wet down and cleaned at the start of the shift, and periodically wet down and cleaned as required thereafter.

All main haulageways and inlet air courses are periodically wet down, different methods being used under different mine conditions. In some places a large sprinkling tank is used; in others, permanently fixed sprays; and on some levels a hose is carried by a man, whose sole duty it is to wet down certain areas. Fixed hose are also used at other places.

Tops of all raises in active zones are provided with short sprinkling hose. Main-raise travelways are provided with sprinkling hose at all offsets, or else atomizing water sprays handling small quantities of water.

A general rule has been made requiring the wetting down of all working places at the start of the shift before any other work is done, and special sprinkling hose are provided for the purpose.

All dead-end places are equipped with blower and flexible tubing for auxiliary ventilation, and the end of the flexible tubing must not be more than 25 feet from the face. In drifts and crosscuts, the air is discharged from an oblong discharge piece, which causes it to flow in a tabular stream that does not impinge on the men's bodies but strikes the mine back slightly ahead of the breast. The air stream then follows down the face and out along the floor.

In raises special precautions have been found necessary. Chute sides next to the manway are solidly lagged or bricked. Spaced bricking next to the manway has been eliminated. An individual blower, with flexible tubing, is installed for each raise, and the tubing must be carried up to within 8 feet of the back. Grated safety landings are used to facilitate ventilation. In three-compartment raises, the compressed air-water blasts have two discharge openings.

As a result of research work it was found that the dust disseminated into the air during drilling might be greatly reduced by increasing the quantity of water forced through the drill steel to the cutting edges of the bit. Changes in all stoper drills have been made, therefore, which will approximately double the flow of water through the steel.

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Wetting agents. - Owings (40) has called attention to the fact that fine, dry coal dust may take on "floating" qualities, and the exceptionally fine dust may not follow laws of gravity in all respects, because the proportion of surface to mass is great. When a particle of dust is wetted, its mass increases, it usually agglomerates or flocculates, and it tends to fall. Therefore, water with or without a wetting agent is generally used in dust-allaying methods; but coal is difficult to wet with water, owing to the fact that it does not readily break down the high surface tension of the water. Consequently, fine particles of coal often are seen floating on water; moreover, the finer the particles of dust, the more difficult it is to wet them, as the surface area is greater proportionately, and the finer the particle of water, the greater is its ability to wet the coal dust. Some coal dust apparently is more difficult to wet than others. This may possibly be due to a higher content of resin or other extraneous matter in some coals, which may increase the interfacial tension between the coal and the water.

The surface tension of water may be lowered to a large degree by using suitable wetting agents. Then the droplets of the wetting solution, instead of assuming a globular form, spread over the surface of the dust particles, forming a thin film. When the dust particle is wetted, it falls out of suspension much more rapidly than when dry or partly wetted with water. It has been found that with suitable wetting agents in the water, dust in suspension may be decreased as much as 80 percent over the amount in suspension when water is used to allay the dust, and under such conditions only about one-third to one-half as much water need be used.

Owings (40) summarizes some of the dust-control measures that employ water and wetting agents, as follows:

Bituminous-coal dust may be allayed by applying water alone or water to which a suitable wetting agent has been added. Dust should be allayed at its source and before it is thrown into suspension whenever possible.

Water may be piped into the mine and to the working faces; it may be taken into the sections in portable tanks, or it may be supplied directly to individual pieces of equipment from tanks.

Plunger pumps are satisfactory for main pipe lines, but all-bronze turbine pumps are best suited for the low-capacity, high-pressure nozzles utilized for individual pieces of equipment or on tanks supplying liquid to equipment in separate sections of the mine. Self-cleaning strainers should be installed on the suction side of pumps to reduce wear by impurities in the water and also to prevent clogging of sprays.

Water alone may control dust in some mines and in some operations in mines, provided the method of applying it is efficient. The amount of water used and the amount of dust in suspension may be reduced at least half in some instances by adding a suitable wetting agent to the water. Reduction of as much as two-thirds in the amount of water and 80 percent in dust in suspension has been achieved by adding a wetting agent to water. Dust in suspension requires a quick-acting wetting agent, but where a wetting solution is applied to coal before it reaches the dust-forming operation, a penetrating wetting agent may be effective if enough time elapses after application to allow the slower-acting ingredients to spread the solution over the dust particles.

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Effective application of water may be obtained only by the right degree of atomization; in general, the finer the spray, the better the result in wetting dust particles. Relatively high pressures are required for fine atomization; for undercutting, pressures of about 60 pounds per square inch are usually effective, but for other operations, pressures of 100 to 120 pounds per square inch are necessary.

Control of dust during undercutting is not particularly difficult. A system for effective dust suppression in undercutting is recommended. Preferably, a narrow-angle spray nozzle should be directed onto the bits near the end of the cutter bar on the intaking bits and also on the outcoming bits. A hollow-cone or solid-cone nozzle should also be directed onto the outcoming bits near the motor housing.

Track-mounted machines that cut above the bottom should be equipped with a narrow-angle nozzle on the intaking-bit side, and one or two narrow-angle nozzles should be directed onto the outcoming bits. If dust cannot be controlled effectively in this way, one or two solid-cone nozzles should be placed several feet back of the chain; then water or a wetting solution may be directed on the machine cuttings as they fall from the cutter bar.

Mechanical loading machines should be provided with two solid-angle nozzles, one on each side of the machines and about 6 feet from the end of the loading head. Under some conditions, however, spray systems may not be required if the machine cuttings have been wetted adequately or if the coal pile has been wetted after blasting. Sprinkling the coal pile with a wetting solution with deep penetrating characteristics may obviate the necessity of spraying during loading.

Solutions may be applied on underground conveyors near the discharge from the face conveyor onto the room conveyor and at the discharge from the room conveyor onto the mother conveyor or into cars. Flat-spray nozzles may be employed for directing liquid onto coal in belts, but solid-cone nozzles are preferable at discharge points.

Dust may largely be eliminated at dumps, in tipples, and in cleaning plants if a penetrating wetting agent is applied on the room or entry conveyors or on top of loaded cars either at the belt discharge point or at sidetracks. Flat sprays should be installed on conveyors and pipes with holes drilled in them, or high-capacity solid-cone nozzles actuated by a quick-acting valve may be installed on sidetracks.

The equipment required for underground dust control consists of a self-cleaning strainer, a pump capable of delivering small quantities of water at high pressures, a cut-off valve in the machine, another self-cleaning strainer in the pipe or hose system on the individual piece of equipment, and suitable nozzles.

At mine-car dumps, flat-spray nozzles supplemented with solid-cone nozzles in corners should be so installed that the coal will be covered with a sheet of spray as it is dumped. In most cases, nozzles should be spaced about 18 inches apart around the perimeter of the enclosure. Sprays directed onto conveyors in tipples or cleaning plants should have a flat pattern, but solid-cone sprays should be provided at discharge points.

Flat sprays should be installed above and below crushers in horizontal planes and about 18 inches apart in these planes.

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Wetting agents may be added to water in bulk by pouring the liquid or powder into a tank. Some wetting agents are readily miscible, others are miscible if placed in a tank before the water is added, or pumps may be required to form a suitable mixture. Readily miscible liquid wetting agents may be added where and as needed by utilizing a proportioning feeder. The eductor or syphon-ejector feeder, where it may be used, probably will give little trouble, as it has no moving parts. To obtain constant results, the rate of discharge should be fixed within reasonable limits.

The Bureau of Mines has no established maximum concentration for the number of particles of coal dust in air, but tentatively it is suggested that concentrations should not exceed 20 million particles of dust per cubic foot of air. This limit is now being maintained in some mines during undercutting. The goal undoubtedly can be attained in other operations if attention is given to solving the dust-control problem. Higher concentrations of dust in air may cause dangerous quantities of coal dust to be deposited on exposed surfaces, increasing the explosion hazard, and the occurrence of high concentrations of dust in air may cause harm to the health of workers as well as adversely affect safety in the places in which dustiness is excessive.

Alert coal-mining companies are now taking steps to reduce air dustiness to a minimum, not only to protect themselves against litigation but also in the interest of increasing the efficiency of workers and the equipment with which they work.

An increase of approximately 10 percent in the efficiency of workers is claimed where water is used to allay dust in some mechanized mines in the Midwestern States (36); much of this is probably due to better visibility, but other important factors are the lessened discomfort when air-borne dust is reduced and eyes, ears, nose, throat, and lungs are freed of the hazards that accompany the dust. Preliminary tests indicate an average increase of about 15 percent in illumination through the use of water to allay dust; other tests, which the authors did not conduct, are reported to have shown a decrease in visibility up to 75 percent or more owing to dust in the air of mines in which cutting was done dry. The clearer atmosphere in mines employing water in the various mining operations unquestionably improves the morale of workmen; they are likely to be more contented with working conditions, and as a consequence labor turn-over probably will be reduced. If a decrease of 10 percent is assumed in labor turn-over in normal times owing to watering in coal mines, the cost of installing watering systems in many mines will be more than offset by the reduced cost of employing and training new employees.

A wet method, known as water infusion, is being used successfully in some of the coal mines of Great Britain and some European countries. By this method, water is introduced into the coal seam in advance of the working face, so that the coal is already damp when it is cut. It is said (41) to be a most effective method, but is not free from technical difficulties, particularly if the water seeps from the coal bed into the adjacent strata, which it may soften and thus cause difficulties in the control of the roof.

The water infusion method has been used with some success in some of the coal mines of Belgium and is continuing to be developed in spite of some failures in hard, compact coal. Three tests of systematic control made by the Institute of Hygiene of Mines of Belgium (42) have shown improvement of 40 to 70 percent. The

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tests revealed that the efficacy of the method depends a great deal on the conditions of the deposit and the manner in which the injection is carried out. Homogenous veins, friable and fractured, present the most favorable conditions, particularly if the surrounding terrain is good. A bed with barren inclusions, a very weak opening, or a bad roof are so many conditions making the application of the infusion much more difficult.

From the point of view of the operating method, two factors were found (42) essential - control of the quantity of water and limitation of the speed of injection. The utilization of excessive quantities of water can lead to recession of the roof or weakening of the walls. On the other hand, too rapid injection injures by penetration of the water into the cracks and reduces the efficacy of the method. In practice, a quantity of water around 1 percent of tonnage product generally was sufficient for efficient use of the method.

Routine dust precautions on the Witwatersrand (43) comprise: Wet drilling; the wetting of all materials at blasting and during mucking operations; the washing and sweeping of traveling ways; the provision of dust extractors at chutes, tips, filling points, and crushing machinery, and exhaust fans operating from all development ends. At tipping points, a downdraft of at least 100 feet a minute over the grizzley area is considered necessary in order to prevent upsurges.

Sawdust and flannel filters are used in dust extractors and are of about equal popularity. "Lorna Doone" flannel is used; on some mines it is treated with zinc sulfate to prevent rot by molds. The flannel is made up into long cylindrical bags nested in parallel in a chamber. Air velocities through the flannel are adjusted to 20 to 30 feet a minute. This gives a filtering efficiency of over 90 percent. Bags are cleaned every 10 to 20 days.

Sawdust filters are constructed with two beds of sawdust in series. Each bed is 2 to 4 inches thick, and the air passes down through it. The sawdust is usually raked over periodically and changed every 2 or 3 weeks. Efficiencies are also over 90 percent, except for a few hours after raking (43).

CONTROL OF THE PHYSICAL QUALITY OF MINE AIR

Need For Control

As the workings become deeper, the air in many mines, especially metal mines, becomes increasingly warmer, until working conditions become intolerably hot.

Sources of Heat in Mines

The main sources of heat underground are listed by Jeppe (15) as:

1. Heat due to adiabatic auto-compression of the air in passing down into a mine.

2. Heat from the rock surrounding the air passages. This is dependent mainly on the factors of the rock temperature gradient, the physical properties of the rock, and the "emissivity."

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3. Heat created by men, machinery, and lighting.

4. Heat caused by the oxidation of the mine rock or its constituents, or by the oxidation of timber, etc.

5. Heat caused by blasting.

6. Heat caused by rock movements.

7. Heat caused by friction of the air currents.

Adiabatic compression. - When air flows down a shaft into a mine, it is compressed at the rate of about 1.01 inches of mercury per 1,000 feet of descent, and the air is heated in the same way as if it were compressed in a compressor. If there is no heat interchange between the air and the shafts and airways and no evaporation of moisture takes place, the air increases in temperature at the rate of about 5.4° F. per 1,000 feet, or at the rate of 1° per 185 feet of descent. Thus, the temperature-rise due to compression alone in workings 4,000 feet below the surface would be of the order of 21.6° F. As the wet-bulb temperature rises about 1° for every 2° on the dry bulb with adiabatic compression, then the corresponding increase in the wet-bulb temperature will be approximately 11° F. (44).

The rate of increase in temperature with increase in depth is nearly constant and is independent of depth, temperature, or pressure. The rate of increase of pressure with increase of depth is a function of gravity; hence, the rate of increase of temperature with increase of depth is also a function of gravity (15).

In the Witwatersrand mines (15), adiabatic compression generates in the air about 500,000 B.t.u. per 100,000 feet depth of shaft. It is one of the most important sources of heat in underground air and is unavoidable (15).

In hot mines of the Witwatersrand (45) the heat content of the air is increased nearly 4 B.t.u. per pound by adiabatic compression, the pressure at a depth of 6,000 feet being nearly 3 pounds per square inch higher than at the shaft collar.

Heat from rock surrounding the air passages. - The temperature of the rocks forming the crust of the earth rises with increase in depth to an extent that varies greatly with different types of rock, depending on their age and on the physical factors, such as conductivity and specific gravity, rate of heat flow through the void, and on a number of other factors, such as the presence of dykes, the existence of sulfides in the rock, and so on.

Observations of rock-temperature gradients in the Central Witwatersrand have given results varying between 1° F. rise per 167 feet increase in depth to 1° F. rise for 236 feet increase in depth, in each case the figure being a mean of a number of such observations.

In the Anaconda copper mines, the rise of temperature with depth - "the rock-temperature gradient" - is about 1° F. per 30 feet increase in depth.

In Great Britain the rock-temperature gradient ranges from 1° F. per 42.3 feet to 1° F. per 106 feet increase in depth, with a general average of about 1° F. for

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every 71 feet. In the Hollinger-McIntyre gold field, the rock-temperature gradient is about 1° F. for 223 feet increase in depth.

The base point from which the temperature rise is calculated is taken at 50 feet (some authorities prefer 100 feet) below the surface, where, investigation has shown, the rock temperature remains about constant throughout the year at the average surface (dry-bulb) temperature. At this point, therefore, the temperature varies greatly with different climates.

The temperature at 50 feet within the earth in the Canadian gold-mining districts is about 40° F., whereas in the Kolar gold-mining district it is about 83° F. This wide difference in initial temperatures is of great significance. Thus, for the same rock-temperature gradient the temperature at any depth in the Canadian mines would be some 40° F. lower than that in the Kolar gold field.

Rock temperatures on the deeper levels of the Magma mine of the Magma Copper Co. in Arizona (46) are given in table 5.

TABLE 5. - Rock temperatures at various levels in Magma mine

Level	Temperature, ° F.
3,200	127
3,400	130
3,600	134
3,800	137
4,000	140
4,200	143
4,400	146
4,600	150

The geothermal gradient, as indicated by the rock temperatures given in table 5, is 1-1/2° per 100 feet.

Heat created by man. - The chemical changes that take place in the human body produce heat. Even when a man is at rest, the basal metabolism creates an appreciable amount of heat. An ordinarily clothed man sitting quietly in a temperature of 60° F. and 50 percent relative humidity is said (15) to produce 1.2 large calories of heat a minute (238 B.t.u. an hour). A man doing extremely hard work may produce up to 10 times as much heat as when he is at rest (up to 2,500 to 3,000 B.t.u. an hour).

A miner working at his full normal capacity may produce heat at the average rate of about 250 large calories an hour or 1,000 B.t.u. an hour (43). Assuming that there are 600 miners underground working at an average rate, the heat-output will be about 600,000 B.t.u. an hour. Taking the volume of air flowing through the mine at 300,000 cubic feet per minute, the temperature-rise equivalent to this access of heat would be about 2° F., which is, of course, very small. Actually, in a hot mine, where body-temperature is less than the air-temperature, the heat loss from the body would be by evaporation of sweat, and the wet-bulb temperature would be increased, not the dry-bulb temperature.

Lawton (43) has recorded a case of a hot mine in which 20 men were employed on a face 75 yards long, along which 3,700 cubic feet of air was passing a minute. He

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found that when the men were off the face, the wet-bulb temperature was 4.3° F. lower than when they were working. The whole of this rise in wet-bulb temperature may not be due entirely to sweating of the men, as moisture also may be evaporated from the freshly-exposed coal surfaces produced by the men when working.

Heat created by machinery. - The net amount of heat given to the air as a result of the presence of power-operated machines in coal mines depends on the type of power used and on the kind of work done by the machine. The usual power systems employed in mines are electricity and compressed air. In level workings, most of the energy put into underground machinery is absorbed in overcoming frictional loads and therefore eventually enters the air in the form of heat. Work done against gravity, however, as in pumps raising water to the surface and haulages raising coal from dips, increases the potential energy of the substance raised, and this portion of the energy input does not appear as heat.

All losses in electrical machines, such as iron and copper losses in transformers and motors and the power losses in cables, are transformed into heat. Most of the energy given to haulage engines, coal cutters, and conveyors underground eventually passes into the air as heat, as it is utilized mainly in overcoming frictional loads. According to Hinsley (44), with electrical power systems, almost the whole of the power introduced into the mine eventually enters the air as heat and therefore raises the air-temperature in proportion to the amount of power used. The magnitude of the temperature-rise in the average mine from this source will not be very large.

With the large-scale operations on the Witwatersrand mines, a very large amount of power is used, and since the machines work on the average at considerably less than 100 percent efficiency, large quantities of heat are created and pass into the air.

A 2,000-h.p. hoist underground may put 1,000 h.p. into the air in the form of heat (1 h.p. = 42.72 B.t.u. in an hour).

The work done by fans is almost entirely converted into heat. The temperature rise due to adiabatic compression (due to the water gage difference of pressure induced by the fan) is about 0.375° F. per 1.0 inch water gage. This may be an important factor if the fans be placed in the downcast shaft or immediately in front of or near to the working faces. Hinsley (15) states: "The only portion (of the work done by the fan) not appearing as heat is that small amount represented by the increase in kinetic energy due to the action of the fan."

The energy losses in the driving motor, the driving mechanism, and the aerodynamic losses in the fan are turned almost immediately into heat and pass into the air (44). The compression of the air in the fan through the water-gage difference of pressure also increases the heat content, the temperature rise for 1 inch of water gage by adiabatic compression being approximately 0.4° F. Thus, if a booster fan were installed underground, which required 100 h.p. to drive it, and the over-all efficiency of the installation were 50 percent, then the heat equivalent of 50 h.p. enters the air as losses. The other 50 h.p. is used in compressing the air, thereby still further raising its temperature and giving it kinetic energy. The air as it flows to and from the fan encounters friction, and mechanical energy of flow is converted into heat. At the same time a pressure-drop expansion takes place, and the heating effect of friction is almost exactly counterbalanced by the cooling produced

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by the pressure loss. It will be realized, therefore, that almost the whole of the 100 h.p. put into the fan motor eventually appears as heat in the air. The only portion not appearing as heat is that small amount represented by the increase in kinetic energy due to the action of the fan. The fan placed in an intake airway and passing 60,000 cubic feet of air a minute would raise the temperature of the air by about 4° F. and the wet-bulb temperature by about 1.5° F. By placing the fan on the return side of the coal face, this accession of heat is prevented from being passed on to the coal face, and the main effect of the fan upon the workers is then one of cooling due to the increased air velocity.

The limit to the use of intake fans is reached when the cooling effect felt by the men due to increased air speed is just balanced by the warming effect due to the access of heat from the fan and its motor. The higher the over-all efficiency of the installation, the less is the heat put into the air for a given increase in the air quantity.

The heat produced by lights may be significant, especially if carbide lights are used. The generation of heat factor is of particular importance in hot, deep mines and may be sufficient in itself to determine the form of illumination. Thus, by replacing an installation of 5,000 acetylene lamps by 5,000 electric cap lamps (three-cell), the quantity of heat produced would be reduced from 1,165,000 B.t.u. an hour to 12,500 B.t.u. an hour, that is, by 1,152,500 B.t.u. an hour, which is equivalent to a cooling of 92 refrigeration tons (15).

Heat due to oxidation. - In coal mines, 80 to 85 percent of the heat put into the air may come from the coal; the amount depends on the specific heat of the coal and its water content. In sulfide mines such as those at Rio Tinto in Spain, heat from oxidation may be very great. Thus, Brown (15) reports that at the bottom of the upcast shaft, almost 7,000 feet from surface, the temperature of the air was 92° F. - nearly 30° above the probable rock temperature.

In metal mines, including gold mines, where some sulfide is present, the oxidation of pyrite may raise the temperature of the air above the rock temperature (15).

On the Witwatersrand, the 2 to 3 percent of pyrites normally present in the "banket" may have a small heating effect, though some authorities (Rees) hold that it is important as the heat from the rock. In any case it is included, in practice, in the heat from the rock.

Haldane (15) pointed out that in coal mines the heat produced by oxidation can be estimated from the analysis of the intake and return air. Heat from oxidation of timber is often appreciable. Not uncommonly on the Witwatersrand, in heavy timbered haulages, when large currents of air are not passing, the temperature of the air may be several degrees above that of the virgin rock temperature.

Heat caused by blasting. - The heat caused by blasting is very great. Ewing and Egan (15) estimate the heat produced by blasting in a mine milling 100,000 tons a month as about 0.3 million B.t.u. an hour, taken as an average over the whole month.

Assuming that blasting is concentrated in a single hour, on one shift a day, the relative figure would be about 5 to 6 million B.t.u. for that hour.

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This heat, however, is created when the majority of the underground labor force is out of the mine and normally is dissipated before the next shift goes underground.

Heat caused by rock movements. - The heat produced by large rock movements is susceptible to assessment from the factors of the weight involved, the distance of movement, and the friction factor.

Moss (15), making certain assumptions, concluded that in coal mines rock movements may theoretically give about 9 percent of the total heat put into the ventilating air, but that in practice it is only a small fraction of 1 percent.

The heat created is rapidly dissipated, partly in the air currents, but mainly, it would seem, in the rock masses. The effect of large numbers of small movements (such as is not uncommon in the older mines of the Witwatersrand) may be more important than is generally realized.

Heat caused by friction of air. - The heat caused by friction of air against the walls of the airways may be considered negligible. The air is heated by friction, but the friction causes a drop in pressure and thus an expansion of the air. The cooling by this expansion probably balances the heat produced by the friction (15).

Effect of Heat and Humidity on Working Capacity

Studies have been made in various parts of the world to determine the effect on working capacity of miners in deep hot mines. Studies made by the Federal Bureau of Mines (47) in comparatively deep copper mines, with fairly high temperatures and humidities, revealed that the efficiency of the workers was much less than 50 percent that of similar workers in other mines. At working faces, while one machine man or mucker worked his companion rested in the full stream of a compressed-air blower, the men exchanging places at intervals of 20 to 30 minutes, and frequently both rested. Moreover, the man working the short interval at the face worked at reduced blower pressure; for instance, two men at the face of a drift in this mine in still air, with 96° F. wet bulb and 94 percent relative humidity, mucked about 12 tons per shift, whereas in a drift in an adjoining mine less than 1,000 feet away, in moving air, with 82° F. wet bulb and 82 percent relative humidity, two men mucked 30 tons or over per shift. The average of about 30 readings taken at all working faces of this mine gave wet bulb 93.3° F., dry bulb 94.4° F., and relative humidity 96 percent, and at no place was there a perceptible movement of air except at points close to compressed-air blowers. The monthly labor turnover was 100 percent.

If temperatures are such that a man's body temperature continues to rise during work, he automatically cuts down his rate of working to relieve the discomfort produced. A difficulty encountered in tests on working capacity is that of finding a suitable measure of work output for coal miners, as working conditions in mines vary considerably. The output of coal per man-shift is not a good measure, as coal production varies considerably and may be easier in the deeper and hotter places.

Vernon, Bedford, and Warner (48) employed two other tests in investigations on the relation of warmth to working capacity, the time taken to fill a 10 cwt. of coal, and the time taken by the miners in short rests. They found that in the hotter places both the time taken from resting and the time required to load 10 cwt. of coal increased. The following table shows the results of these tests as arranged by Yaglou (48):

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TABLE 6. - Relation of warmth to working capacity

Effective temperature, °F.	Wet Kata cooling power	Rest pauses, minutes per hour	Tub-filling time, minutes	Relative output, percent
65.8	18.6	7.3	8.0	100
65.8	14.6	6.7	8.6	94
75.3	12.9	9.0	8.5	91
75.8	10.8	10.0	9.2	82
77.8	9.0	11.1	9.1	81
81.8	6.4	22.4	9.6	59

Tests made at the Armored Medical Research Laboratory (49), at Fort Knox, Ky., on soldiers who had been acclimatized to hot conditions, both dry (desert) and humid (tropical) heat, confirmed observations made in industry and the original conclusions of Haldane (50) that the wet-bulb temperature of the environment is the most important factor in determining the upper limits of heat and humidity at which men can work.

The tests at the laboratory led to the following conclusions from the study in which the dry-bulb temperature ranged between 93° F. and 121° F. and the wet-bulb temperature between 90° F. and 96° F.:

1. Within the range of environments here studied, the wet-bulb temperature is the limiting factor, which determines the ability of men to work in hot environments. The dry-bulb temperature exerts only a minor influence in this regard.

2. As the upper environmental limits are approached, a narrow range of wet-bulb temperature, 4° F. to 5° F., separates environments in which work is relatively easy from those in which work is impossible.

3. When the wet-bulb temperature is below 91° F., men work easily, efficiently, and with only mild physiologic changes.

4. At wet-bulb temperatures between 91° F. and 94° F., men are capable of prolonged, moderately hard work, but they now work inefficiently and with difficulty, lose vigor and alertness, sustain undesirable physiologic changes, and may become mild heat casualties.

5. Moderately hard work at wet-bulb temperatures of 94° F. and higher leads rapidly to total disability in most men, with excessive and often disturbing physiologic changes. Approximately 1 hour of sustained work is tolerated by most men; those who work longer do so inefficiently and with marked physiologic disturbances.

6. The limiting wet-bulb temperatures for each of the above three types of performance (3, 4, 5) are slightly (approximately 2° F.) higher at dry-bulb temperatures below 100.0° F. than at dry-bulb temperature of 120° F.

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	Relatively easy	Difficult	Impossible
Completely saturated environment	92.5	94	96
120° F. dry-bulb temperature	90.	92	94

7. At the upper environmental limits, sweating is extremely profuse; most men average 2.25 liters per hour, and some reach 3.5 liters per hour.

8. Acclimatized men working above the upper environmental limits develop marked physiologic changes and undesirable, frequently disabling, symptoms similar to those sustained by unacclimatized men when first working in the heat.

According to Hinsley (51), the conclusion that limiting conditions in hot environments could be stated in terms of the wet-bulb temperature and that the dry-bulb temperature was relatively unimportant did not appear to agree with the basic effective temperature chart issued by the American Society of Heating and Ventilating Engineers. From that chart, the dry-bulb temperature appeared to have appreciable effect upon the warmth of environmental conditions. For example, it might be shown that, by changing from conditions represented by 90° F. saturated in still air to 120° F. dry-bulb and 90° F. wet-bulb temperature in still air, the effective temperature was raised 6°. Thus, according to the effective temperature scale, this 30° elevation of the dry-bulb temperature had been equivalent to raising the wet-bulb temperature from 90° to 96°.

Effective temperature is a measure of the air conditions that affect the human feelings and gives in one figure the resultant of the dry-bulb temperature, the humidity, and the velocity of the air. It is based on the fact that many different combinations of temperature, humidity, and air velocity provide the same feeling of degree of warmth.

Combinations of temperature, humidity, and air velocity that induce the same feeling of warmth are called thermo-equivalent conditions. A series of tests carried out at the American Society of Heating and Ventilating Engineers Research Laboratory have established these thermo-equivalent conditions, which are claimed to indicate both the sensation of warmth and the physiological effects on the body induced by heat or cold. For this reason, the resulting temperature scale was called the effective temperature (52).

Effective temperatures cannot be measured on a thermometer but are obtained from tables or specially drawn charts. The numerical value of the scale was fixed for convenience at the value where the wet- and dry-bulb temperatures agree, that is, at saturation. Thus, any air condition has an effective temperature of 70°, for instance, when it induces a sensation of warmth like that experienced in calm air at 70° saturated with moisture.

Two charts of effective temperature have been prepared - the basic scale applicable to persons stripped to the waist and the normal scale applicable to persons

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wearing customary indoor clothing. The basic scale is the one that is applicable to conditions in hot mines.

The effective temperature scale is much used in the United States as a measure of the comfort of different air conditions (temperature, humidity, and velocity), and in 1944 Hinsley (15) made a close study of the scale. He found no significant relationship between the wet kata and the effective temperature scales, but came to the following conclusion:

"From the experience gained during this inquiry, the conclusion has been reached that in hot mines the effective temperature is a far more satisfactory measure of environmental warmth than is the wet kata cooling power."

The Royal Commission on Safety in Coal Mines, in the Third Report of the General Research Committee (1942), states:

"Neither kata cooling power nor effective temperature is an adequate measure of conditions by itself, but the two methods are complementary and can be used together to give a clearer picture of conditions than either can be separately."

Jeppe (15) discusses the effective temperature scale and the wet cooling power as follows:

> For effective temperature determinations, dry-bulb kata readings are necessary; it is easy to take wet-kata readings at the same time.
>
> The wet kata overestimates the effect of the air movement and underestimates the effects of temperature and humidity; it therefore does not give a true representation of the effect of a given environment on the human body - a most important matter in mining because of the great range of temperatures encountered.
>
> On the other hand, the very sensitivity to air movement makes the wet kata an excellent index of ventilation conditions.
>
> The effective temperature scale is a much truer index of warmth, but except at high temperatures, it is possible to comply with an effective temperature scale without recourse to air movement, i.e., it does not demand that degree of ventilation which is so necessary to maintain freshness.
>
> Thus, the wet kata shows the advantage as an indicator of the fitness of ventilation conditions, whereas the effective temperature scale is the more reliable measure of warmth. At extreme conditions of temperature, the effective temperature scale is superior to the wet kata.
>
> Attempts to plot these two quantities against each other, and both of them against velocity, dry kata cooling power and high temperature cooling power emphasize the extreme difficulty of establishing relationship between the various factors concerned.

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The criticism commonly directed against the effective temperature scale is that:

1. Sensations of warmth and cold are not, in general, very reliable. They are dependent on the individual and on immediate past experience.

2. Opinions of a majority of persons, even when the number of subjects is large, cannot be accepted with such confidence as results given by scientifically evolved instruments.

It is certain, however, that such factors have been taken into consideration by the investigators and due allowance made for them, and the effective temperature scale must be accepted as one means of assessing the cooling power of air.

As shallow deposits become exhausted or present ore bodies are followed to greater depths, the problem of working efficiency at high temperatures is becoming of increasing importance to many mining fields. It is of general interest, therefore, to present in some detail the investigations carried out by Caplan and Lindsay (53). They attempted to determine a satisfactory index that could be adopted as a measurement of the comfort and efficiency of workers in deep mines and the point at which a serious falling off in efficiency might be expected.

They felt that more reliable results would be obtained if the men (Indians on the Kolar Gold Field) used as subjects performed work to which they were accustomed in their daily routine. Six men of average physical fitness, daily engaged in drilling rock by hand and acclimatized to hot and humid conditions up to 93° F. wet-bulb temperature, were selected. Complete data were obtained on 71 tests. Following are the main facts brought out by the investigation:

1. During a 3-hour shift the output of work diminished hourly even under the most comfortable working conditions. This hourly decline in efficiency was increasingly manifest when the wet-bulb temperature rose above 83° F., the effective temperature above 85°, and the wet kata cooling power fell below 14. Ultimately, the efficiency fell to a degree when little or no useful work could be performed in the third and even the second hour. This stage was reached when the efficiency fell below 25 percent and was noted:

   a. After 2 hours' work, when the wet-bulb temperature rose above 94° F., the effective temperature above 97°, and the wet kata fell below 3.

   b. After 1 hour, when the wet-bulb temperature rose above 95.5° F., the effective temperature above 99°, and the wet kata fell below 2.

A serious falling off in efficiency (say 50 percent) occurred, however, at considerably less oppressive conditions. This was noted:

   a. After 2 hours' work, when the wet-bulb temperature rose above 91.5° F., the effective temperature above 94.5°, and the wet kata cooling power fell below 5.

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b. After 1 hour, when the wet-bulb temperature rose above 93° F., the effective temperature above 96.5°, and the wet-kata cooling power fell below 4.

During the first hour the output was not seriously impaired - fell below 50 percent - until the wet-bulb temperature rose above 94.4° F., the effective temperature above 98.0°, and the wet kata fell below 1.8.

It may be assumed that during a normal 6-hour working shift the efficiency would continue to fall with each successive hour, and after the third hour a serious falling off in output would occur in conditions less oppressive than those found in the present investigation.

2. In a complete 3-hour shift the efficiency began to fall when the wet-bulb temperature rose above 83° F., the effective temperature above 85° F., the wet kata fell below 13.5. When conditions deteriorated, the efficiency fell at first gradually and then more rapidly. If it is assumed that work was (a) moderately impaired when the efficiency was 75 percent, (b) seriously impaired at 50 percent, and (c) useless below 25 percent, these stages were reached:

a. When the wet-bulb temperature rose above 89.5° F., the effective temperature above 91.5°, and the wet kata fell below 7.5.

b. When the wet-bulb temperature rose above 92.5° F., the effective temperature above 96.0°, and the wet kata fell below 4.0.

c. When the wet-bulb temperature rose above 95° F., the effective temperature above 98.5°, and the wet kata fell below 2.

Again, it may be assumed that during a normal 6-hour working shift there would be a greater all-around reduction in efficiency under the conditions described.

Haldane (15) suggested 78° F. as the critical point, and this was later confirmed by Ehrismann and Hasse (54), who estimated that at 78° F. wet bulb the capacity for work had already been reduced to 60 percent of the maximum.

Experience on the Witwatersrand (15) suggests that 60 percent efficiency at 78° F. wet bulb is on the low side for acclimatized natives, and that maximum effort can still be maintained at well above that temperature. Above 80° F. wet bulb there is a fairly rapid decrease in efficiency. Above 88° F. wet bulb the working efficiency invariably decreases rapidly, although this decrease can be offset to an appreciable degree by increasing air velocity. At a temperature of 92° wet bulb, even with a velocity of 200 feet a minute, the efficiency is almost certainly below 50 percent of maximum capacity (15).

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Methods of Control

Cooling Mine Air by Ventilation

The extent to which increased volume of air can be utilized for improving the cooling power of air underground depends upon conditions at the individual mine. The limiting factors, as pointed out by Spalding and Parker (15), are:

1. When the cost of providing new and larger airways has become prohibitive.

2. When the airways can be no further enlarged, and the cost of fan power to force the air against the resistance of the mine has become prohibitive.

3. When, owing to the increasing rock temperature at depth, the temperature of the air circulating through the workings has reached the point when little or no additional bodily comfort is gained by increasing the air velocity (this is dependent, also, on whether dry or wet mining is practiced).

During the last decade or so, a number of older mines (especially in the main Central area of the Witwatersrand), which have been deepening their workings at an average of 130 to 160 feet per year, have greatly improved their ventilation lay-outs and greatly increased the volume of air flowing through their workings, in addition to making the most efficient arrangements possible for air distribution, and it is possible to arrive at some general working basis of air requirements for mines operating under similar conditions (15).

The very considerable change-over in the ventilation lay-out at the East Rand Proprietary Mines (15) was accompanied by the following carefully planned control of the air distribution:

1. Conducting the air to depth by the shortest route and eliminating leakage as far as possible.

2. Controlling the flow of air in the working areas by suitably placed ventilation doors, regulators, and barriers.

3. Installing a series of exhaust fans across the strike to give the required distribution; the volumes of air delivered can be varied by adjustments to the fans.

4. Insuring that there is no short-circuiting at the exhaust fans, e.g., by making use of natural barriers such as faults and dykes.

5. Providing adequate return airways from the delivery side of the exhaust fans to surface; these airways were planned and can be inspected and repaired regularly.

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6. Connecting the exhaust fans to the current stoping areas by a system of large footwall returns; these insure a strong pull on the air just above the stopes.

7. Increasing the ventilation staff.

As far as possible, the principle was followed by having fast-moving air in the intakes and working areas and slow-moving air in the returns.

According to Hill and Ranson (15), the quantity of air required for ventilating deep mines depends on many factors, but it would seem that the aim should be at least 60,000 cubic feet per minute for 1,000 feet of strike.

Various investigators (15) state that although increase in volume is the first and most important method of obtaining, economically, better air conditions, it is not likely by itself, unassociated with other methods of air cooling, to enable mining operations on a large scale to be carried out on the Witwatersrand to a depth much greater than 7,000 to 8,000 feet.

For a number of years there has been much discussion regarding the relative advantages of dry mining as opposed to wet mining. The most important argument against dry mining is that water is an efficient aid in the control of silicosis. No really successful method other than the use of water has been evolved for allaying dust created by drilling and other dusty mining operations. According to the Deep Level Mining Report (55), the system has been successful in the mines of the Witwatersrand and has helped substantially to prevent silicosis. The air, however, is almost completely saturated with moisture, and humid air at high temperature has little cooling effect upon the body. On the other hand, it is claimed (15) that under dry-mining conditions the wet-bulb temperature would be reduced very appreciably, the reduction being great enough to allow mining to be carried to a depth of 12,000 to 13,000 feet without having the air conditions become unbearable.

The report (55) cites as an example the mining method at the Kolar Gold Field, where, on the contrary, all mining is carried out dry. The dust produced by mining operations is deemed not to be dangerous and not to cause silicosis. There is little water in the mines, and such small amounts are brought to the surface out of contact, as far as possible, with the air of the mine. The air in the Kolar mines is dry, and, as it does not evaporate much, the heat flow from the rock is reduced; for the same rock temperatures, the dry-bulb temperature is higher and the wet-bulb temperature is lower than on the Witwatersrand, thus giving the air a greater cooling effect.

Numerous other advantages and disadvantages of the two systems of mining are discussed at length in the literature.

Cooling Mine Air by Refrigeration

When it becomes economically impossible to provide a large enough volume of air to obtain satisfactory underground air conditions, artificial air cooling becomes necessary.

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The Morro Velho mine in Brazil was the first to introduce refrigeration (15). In 1920 an ammonia plant was installed at the surface to cool the downcast air. In 1929 greatly increased depth of working and higher rock temperatures made necessary the installation of an underground refrigeration plant. This plant was installed at the 5,800 level and consisted of two centrifugal machines using "Carrene", a nontoxic, nonflammable refrigerant. An ammonia plant is not practicable underground, owing to the toxic nature of ammonia.

Table 7 gives the downcast air temperatures at 6,550 feet below the surface in the Champion Reef, Kolar Gold Field, before and after installation of cooling plant (56).

TABLE 7. - Air temperatures, 6,550 feet below surface, before and after installation of cooling plant

	Dry bulb, °F.	Wet bulb, °F.	Moisture grains per pound (dry) air	Air volume, c.f.m.
Before cooling plant installed:				
October, 1940	106.5	84.5	118.5	86,600
After cooling plant installed:				
November, 1940	95.9	73.6	70.2	106,900
October, 1941	90.2	69.1	64.9	112,400
October, 1942	84.4	64.8	46.7	120,600
October, 1943	84.2	65.1	48.2	112,200
October, 1944	86.3	65.9	48.9	110,100
October, 1945	83.8	64.1	44.8	109,400

Barenbrug (56) calls attention to the effect upon efficiency of the workers of the conditions at the Ooregum mine after installation of the cooling plant. Before the installation, when conditions of 117° F. dry bulb, 91° F. wet bulb, and 100 feet per minute air velocity were not uncommon in the deeper zones, almost no work could be performed except for relatively short working periods interspersed with rest periods for "cooling off" in more comfortable zones; the effective and wet kata were 1.0 and 4.5, respectively. Even after the installation of a cooling plant, only 900 ft.-lb. per min. of work could be performed when the average temperature conditions were 118° F. dry bulb and 83° F. wet bulb, with effective kata 2.5 and wet kata 10.0. Many cases of heat exhaustion occurred at the Ooregum mine before the cooling plant was installed.

The second deep mine to install a refrigerating plant was the Turf Shaft of the Robinson Deep in South Africa. Instead of an ammonia plant, such as usually is found at mine refrigeration plants on the surface, three Carrier centrifugal refrigeration units were installed. The liquid refrigerant used was Carrene No. 2, $CFCl_3$ (trichlorofluoromethane). As this is the only plant solely of this type used on the surface, it is described in detail by Jeppe (15). Refrigerating plants at a number of other deep mines also are described by him in some detail.

Before installation of the cooling plant at Robinson Deep, a rate of work approximately 1,700 ft.-lb. per min. could be performed under conditions of 90.2° F. dry

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bulb, 87.9° F. wet bulb, and 100 feet per minute air velocity, with effective kata 3.6 and wet kata 6.6. After the installation of the cooling plant, conditions improved to 89.0° F. dry bulb and 86.5° F. wet bulb, 100 feet per minute air velocity, with effective kata 4.0 and wet kata 7.5; work output was 2,000 ft.-lb. per min. (46).

At the time of installation (1935), when working at full capacity, the cooling effect of this plant was equivalent to sending underground 2,000 tons of ice daily. The plant was delivering 320,000 cubic feet of air per minute into the mine at a temperature of 35° F., although the temperature was actually 32° F. when it left the last refrigerating chamber (58).

Several other mines on the Rand in South Africa now have air-conditioning plants.

The first mine air-conditioning system with mechanical refrigeration in the United States was installed in 1937 in the Magma copper mine at Superior, Ariz. The rock temperature at that time in the Magma mine was 10 degrees warmer at the 4,000-foot level than the Robinson Deep at 8,000 feet (58). The first plant was installed at the 3,400-foot level. The plant consists of two centrifugal refrigeration units of 140 tons refrigeration capacity each, one unit cooling the air for the 3,400 level and the other for the 3,600 level. Each unit is designed to cool 30,000 cubic feet of air per minute 10° F. on the wet-bulb temperature (46).

Prior to the installation, the temperatures of the working places on the 3,400 level averaged 90° F. wet-bulb temperature and 58 percent relative humidity. After 4 months of air-conditioned ventilation, the average of all working places was 72° F. wet bulb, 80° F. dry bulb, and 68 percent relative humidity. The maximum benefit was derived from the cooling system after 7 months of operation (February 1938), when, aided by the cooler incoming air, owing to surface temperatures, the average of all working places was 71° F. wet bulb, 81° F. dry bulb, and 62 percent relative humidity. The number of active stopes on the 3,400 and 3,600 levels had been increased until in 1940 approximately 50 percent of the mine production was obtained from these levels. The temperature of all active stopes averaged 74° F. wet bulb and 81° F. dry bulb.

The corresponding temperatures for the 3,600 level were as follows:

Before air-cooling: Wet bulb 93° F., dry bulb 101.5° F. and relative humidity 74 percent.

After 4 months of air-cooling: Wet bulb 78.5° F., dry bulb 88.5° F., and relative humidity 64 percent.

After this time, the temperatures of the 3,600 level did not improve, owing to the fact that the return air from development below the 3,600 level passed through the 3,600 cooling coils. However, when it was possible to send the return air from the 4,000 level to the main exhaust shaft, the average of all working places on the level was 74° F. wet bulb and 79° F. dry bulb.

A new cooling system at the Magma Copper Co., Superior, Ariz. (59), began operation on August 26, 1948, with the first underground unit installed on the 4,200-foot level. A large surface tower cools 1,720 gallons per minute of water from 85° F. to 62° F. when air entering tower precooler coils is 100° F. dry bulb and 70° F. wet bulb.

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Roughly, 573,000 cubic feet per minute of air can be drawn through precooler coils and discharged at the top of the tower. Approximately 1,650 gallons per minute of water can circulate through the precooler coils and go to the top of the tower with 1,720 gallons per minute of mine water that has been heated in the heat exchangers underground for evaporative cooling.

After the system had been in operation for 24 hours, the temperature of the mine air on the 4,200-foot level in the vicinity of the first cooling unit was lowered from a wet bulb of 94° F., dry bulb 111° F., to a wet bulb of 67° F., dry bulb 72° F. The No. 2 unit on the 4,200-foot level began operation on September 4, 1948. The first cooling unit on the 4,400-foot level began operation on December 13, 1948. A second unit will be installed on the 4,400-foot level and two units on the 4,600-foot level, making a total of six underground units to complete the system. Each underground unit is composed of (a) water-to-water heat exchanger, (b) 25,000-cubic feet per minute fan, (c) 215-gallon per minute circulating pump, (d) nine cooling coils, (e) an air washer with sprays, (f) eliminator plates, and (g) a 100-gallon per minute circulating pump.

According to Koerner (60), from a comfort standpoint conditions on the 3,400 and 3,600 levels were better after 6 weeks of air-conditioning than on the 3,200 level after 3 years of ventilation by fans without air-conditioning.

Richardson (61) reported that air conditioning or air cooling for the ventilation of the Butte mines is distinguished from most air-cooling methods mainly by the fact that the cooling effect is derived directly from the atmosphere without the use of a compressor or other artificial refrigeration machinery.

Although the Butte mines are relatively shallow, virgin rock temperatures are high and variable. They are usually about 120° F. at 4,000 feet. At the Mountain Consolidated mine, where the first air-cooling plant was installed, the geothermic gradient has approximated 1° F. per 50 feet of depth (61), compared with 1° F. per 200 feet of depth in the Witwatersrand mines. This increase in temperature with depth of mining makes the maintenance of a satisfactory standard of temperature and humidity in working places the most difficult factor of the ventilation problem.

Fortunately for an economic solution of the problem, the mean annual temperature of the surface atmosphere at Butte is about 40° F., with very low relative humidity in summer, so that there is unlimited natural cooling power at a temperature range adaptable to air-conditioning methods.

Two similar methods of air conditioning are used, as necessitated by service requirements and dependent on the equipment available. In one, which is the more important and has the greater cooling capacity, water is circulated in a closed circuit and, alternately, first absorbs heat from warm mine air and then dissipates it in the surface atmosphere by cooling at a cooling tower. The second general method is to use cold water, derived either from a source within the mine or from the freshwater supply system, to absorb heat from the mine air and then to turn it to waste in the mine drainage system.

The success of both methods depends upon the fact that, because of the greater density and specific heat of water, a unit volume of water will absorb much more heat than would be absorbed by the same volume of air for the same increase in

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temperature. A relatively small pipe line, therefore, will carry enough water to remove as much heat from a mine as would be carried by a volume of air for which a large shaft would be required. Also, the water is not subject to any increase in temperature because of increase of pressure, and by insulation of the pipe lines the heat transferred to it from surrounding warm air may be more closely controlled.

The water (brine in winter) is cooled at the cooling tower and then flows through a pipe line to an underground air-conditioning plant where it absorbs heat from the mine air; it returns through another pipe to the cooling tower. The underground plant is composed, essentially, of a number of coils of small-diameter pipe that form part of the whole pipe system, so that the hydrostatic head is in balance, and power, furnished by a pump on the return pipe line, is required only to overcome frictional resistance to water flow. Cooling of water at the surface cooling tower is naturally more difficult in summer than in winter, and it has been necessary to use a special method of utilizing the exceptionally dry climatic conditions.

In the Butte district, during the summer, the sensible temperature seldom exceeds 90° F. for more than a few hours per day or during more than 2 or 3 weeks' time. Under such a condition the wet bulb usually will be about 60° F., so that the dew point is about 39° F. Frequently the dew point is below the freezing point, even during the month of July.

As it is not possible to cool water below the wet-bulb (evaporation) temperature of air brought into contact with water at a cooling tower, ordinary counter-current circulation would not give satisfactory results during the summer, especially as it is not possible to attain the theoretical limits. However, a system was developed that makes it, theoretically, possible to cool the water to the dew-point temperature of the air instead of the wet-bulb temperature, and this satisfactorily solved the problem (61).

Richardson (62) has stated that possibly the best general measure of accomplishment is registered by the difference between ground temperature and average temperature of mine air under different conditions. The highest ground temperature noted on the 3,200-foot level in freshly opened ground, before the operation of any air-conditioning plants, was 97.5°, and the average condition for all the working places in the mine was temperature 85.5° F. (wet-bulb, 82° F.). On the 4,100-foot level, the highest ground temperature in freshly opened ground was 118° F., and with three air-conditioning plants in operation, the average mine condition for the same month of the year as before was temperature 82.7° F. (wet-bulb, 80.0° F.). The difference between maximum ground temperature and average temperature of the mine air was 12° F. in the first case and 35.3° F. in the second.

An unusually interesting example of the influence of modern air-conditioning upon production is given by Manchester (63) in his book on New World Machines, published in 1946. The statement is made that without modern air-conditioning the war's great demands upon the copper mines could hardly have been met. Once, when new low levels were opened where temperatures run as high as 150° F., it was standard practice to blow air through the tunnels for 3 years or so until they were cool enough to work in. Now, according to Manchester, mechanical refrigeration cools off these sweltering holes in less than a month, and in one of the United States most productive copper mines the thermometer is kept at 90° F. at a level 4,600 feet underground.

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Manchester's statement is confirmed to some extent by Foraker (64), who said it was necessary to drive the 3,200 level and put up raises through the ore to drain and cool the block from the 3,200 level to the 2,800 level almost 4 years before the ground was cool enough to stope. It is estimated that by air-conditioning, the ore from the 3,600 level to the 3,200 level can be ready for stoping in 6 months.

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